DETAILS OF PRACTICAL MINING

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OF

PRACTICAL MINING

COMPILED FROM THE

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BY

THE EDITORIAL STAFF

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PREFACE

In 1912 we published the first "Handbook of Mining Details," the purpose and scope of which were briefly outlined in the preface as follows:

This book is a collection of articles that have appeared in the *Engineering and Mining Journal* during the last two or three years under the general head of "Details of Practical Mining," a department of the Journal that has been appreciated highly by its readers, many of whom have expressed the wish that a collection in book form be made, which has now been done.

In the editing of this volume, the work has been chiefly in the selection of the material and its arrangement in chapters. Now and then it has been possible to excise some paragraphs as being unessential and occasionally the phraseology of some articles has been altered a little, the requirements of preparation for the original weekly publication not always having permitted leisurely consideration, but in the main the articles now presented in this book are as they were given in the pages of the *Engineering and Mining Journal*. However, it has been necessary in a few cases to reduce the size of the engravings.

In making this collection the limitation of space necessitated the rejection of all material that did not pertain to the subjects selected for the chapters of the book, and even so it was necessary to dismiss some of the longer articles pertaining to them, which approached the character of essays rather than being the description and discussion of details. Of course a wealth of contributions pertaining to the arts of ore dressing and metallurgy had to be rejected summarily. The compilation covers the publications in the *Engineering and Mining Journal* from Aug. 7, 1909, to July 1, 1912. If all of the material that appeared in this department of the *Journal* during that period of three years had been used it would have been necessary to make a book of several times the size of this.

No claim is made that this book is a treatise, exhausting its subject, or any part of it. It is simply a handbook that is a more or less random collection of useful information, being just what passes through the pages of the *Engineering and Mining Journal* in the course of a few years. No special attempt to round out any subject has been made, yet it will be found that some subjects are fully treated.

With regard to the authority of what is to be found in these pages: The matter in the main is merely descriptive of what is done. Nevertheless, there is frequently the injection of opinion and advice. A great technical journal is directed by its editor and is shaped by its editorial staff, but it is essentially the product of its contributors. It is a co-operative institution and its pages are a symposium of the experiences and views of many professional men. During the 18 months ending with June 30, 1912, there were 460 contributors to the *Engineering and Mining Journal*, exclusive of the members of the editorial staff, and its regular coadjutors, and its news correspondents. Many of these contributors furnished articles that are now collected in this book. Their articles generally are signed. The unsigned articles are chiefly the work of members of the editorial staff of the *Journal* who have been sent into the field to study mining practice.

The heterogeneous authorship of this book naturally gives rise to some inconsistencies, some differences of opinion and some conflicts in advice. It has seemed to me

best to let these stand just as in the original, since they are often merely the reflection of different conditions prevailing in different parts of the country, and if carefully read, absence of unity in this respect will not be misleading.

The publication of this volume was favorably received and there was a considerable expression of the opinion that a similar book should be issued later. The idea was conceived, therefore, of publishing such a volume every two or three years. The present volume is issued in conformity with that idea. With respect to its nature and scope, scarcely anything remains to be added to what was said in the preface to the first volume. The main plan remains the same and the nature of the material incorporated is similar, but the classification in chapters and subdivisions thereof is more systematic. This book includes matter that appeared in the *Engineering and Mining Journal* between July 1, 1912, and July 1, 1915.

The preparation of this book, like its forerunner, is stated on the title page to have been by the editorial staff of the *Engineering and Mining Journal,* and in fact every member of the staff worked upon the original presentation of the material in the pages of the *Journal.* The selection and arrangement of the material of this book and the revision—which has been carefully and intelligently done—was the work of Lee O. Kellogg, until recently a member of the editorial staff of the *Engineering and Mining Journal.* Mr. Kellogg is also the author of many of the articles appearing in these pages, and to him belongs chiefly the credit for the preparation of this volume.

W. R. INGALLS.

October 1, 1915.

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DETAILS OF PRACTICAL MINING

I

SURFACE PLANT AND OPERATIONS

Buildings—Power Plant—Shop Appliances—Handling Supplies—Steam-shovel Work—Miscellaneous Notes

BUILDINGS

Sectional Portable Camp Buildings (By George S. Rollin).—The design and erection of the buildings for drilling camps are frequently left to the discretion of the foreman and in most cases when a job is completed, the buildings are a total loss, not being of a type which can be moved. In order to be able to use such buildings over again, plans for portable sectional buildings here illustrated were drawn in the case of a large exploration company. By the use of such standard sections, buildings of any necessary dimensions and proportions can be had; they can be readily assembled and torn down; they become, in fact, part of the regular drilling equipment, shipped with the machinery and returned therewith to the central warehouse. The net result is a considerable saving in time and material. The chief buildings to be provided for are the bunkhouse, the eating house, the drilling shed, and the diamond-setting shed. The walls, roofs and floors are divided into panels made of 2 × 4-in. framing with 1-in. sheathing. The height and width are thus determined; the length can be varied to suit conditions. Special sections provide for doors, windows, corners, ends, etc. The whole building is covered with a tar felt when erected.

The principal building, Fig. 1, is made 20 ft. wide with an 8-ft. wall. Into this there enter eight different sections. In case new sections have to be made, the man in charge of the establishment of the camp is given an erecting sheet shown at 9, of which one is prepared for each of the four types of buildings mentioned. This shows the layout of the building and by its aid the necessary sections can be ordered and all supplies and materials specified exactly.

The instructions which accompany the erecting sheet are as follows:

Determine first the number and position of doors and windows desired. Mark their sections on the erecting sheet by the proper number, thus indicating where the sections will appear in the assembled building.

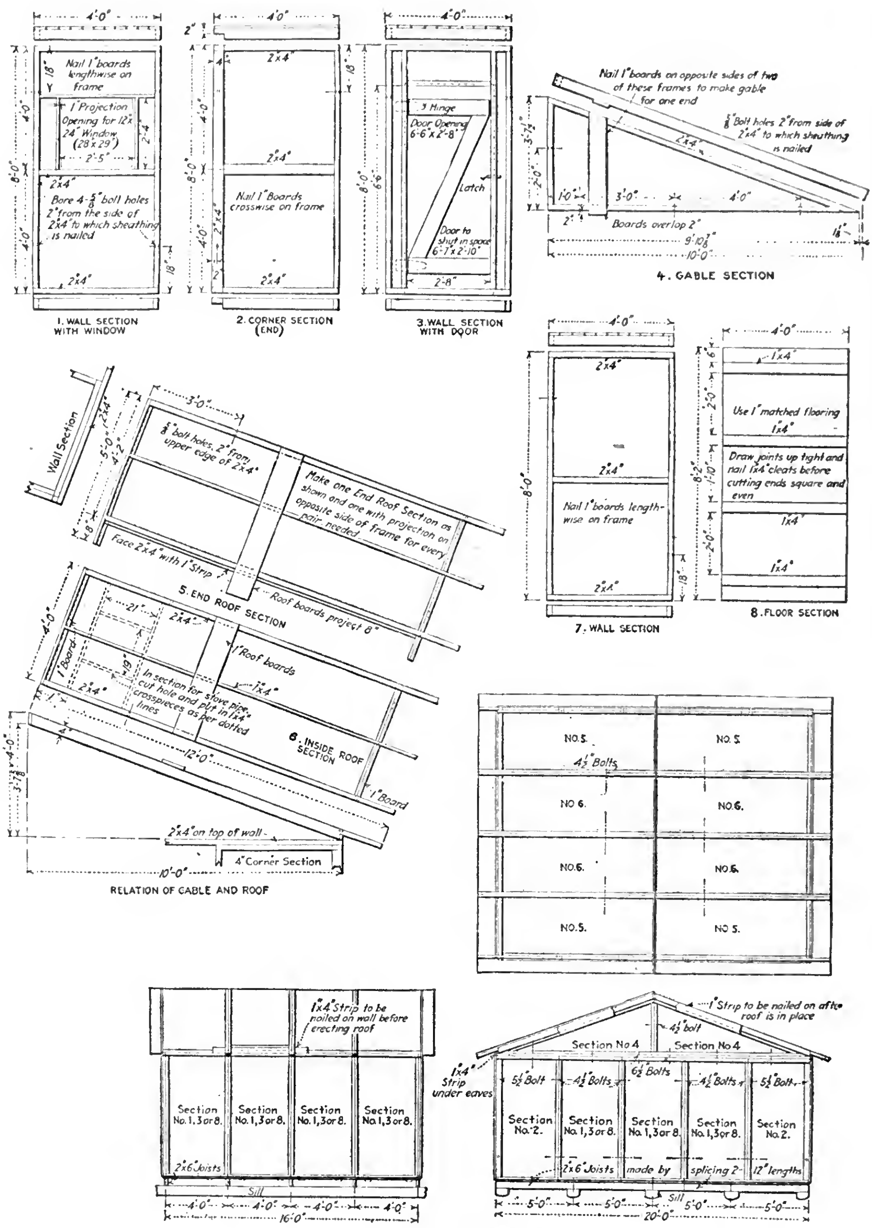


FIG. 1.—SECTIONAL CAMP BUILDING, PANELS SEPARATE AND ASSEMBLED.

The sheet will then show the number of plain wall sections, the window sections, the door sections and the corner end sections entering into the building. In the same way determine and mark on the sheet the end roof sections and inside roof sections, designating those to be provided with stove-pipe holes, and the floor and gable sections. You can now turn over to the carpenter the construction sheets, 1 to 8, and specify how many of each section are required. The erecting sheet should accompany these detail sheets as an aid to the carpenter.

To erect the building: Cut first the five sills from neighboring trees and flatten them on one side at least; lay them lengthwise of the building and level the flat sides with a spirit level. Splice 2 × 6-in. by 12-ft. pieces for the floor beams and lay them across the sills as shown, leveling both ways. Lay the floor sections, 8, lengthwise of the camp and nail them with a few short nails, preferably finishing nails, at each end. This completes the floor.

Set up the wall sections, 1, 2, 3 and 7, in their proper positions as outlined on the erecting sheet. Bolt them together in all cases; if bolts are not at hand, wait for them, and never nail. Square up the walls. Put stays across between the walls at every second section to keep the walls from spreading when the roof is put on. For this purpose, use 2 × 4-in. by 20-ft. pieces, or 2 × 6-in. by 12-ft. pieces spliced. The stays may be fastened to the wall sections just below the top by spiking with 40-penny nails, three to each end. Bind the tops of the four sides with 2 × 4-in. pieces. If necessary to break these along the side walls, see that the break comes over the center of a wall panel. See that the binding 2 × 4-in. pieces come exactly over the 2 × 4-in. pieces forming the tops of the wall panels. Fasten with two 40-penny nails spiked to each panel. The ends must measure 20 ft. over all, in order that the roof section may fit easily.

The gable end sections, 4, should be next erected. They are to be bolted on with 6½-in. bolts; for this purpose, bolt-holes will have to be bored through the 2 × 4-in. pieces forming the wall top, so as to correspond with the bolt-holes already in the gable-bottom piece. Next nail along the top of the side wall the 1 × 4-in. strip, shown in 9, under the eaves and so marked; it should be of good material and should project about 2-in. above the 2 × 4-in. pieces so as to serve for the lower crosspieces of the roof sections to butt against. It may be necessary to plane off a little of the top of this strip so that the lower crosspieces of the roof sections will rest on the side walls and not be held up by the projecting portions which form the eaves, resting on the 1 × 4-in. strip.

The roof is next to be put on. Begin at one end, raising two opposite end sections, 5. The 1-in. board used to face the outside 2 × 4-in.

piece of the roof section should be set flush with the outside boards of the gable section and then the inside edge of the roof section will be flush with the edge of the side wall section on which the lower end rests. In this way the roof and wall sections will correspond for the length of the building. The inside roof sections, 6, are next raised in pairs and are bolted together on the sides with the one bolt, as shown. Care must be taken to get the stove-pipe section in the right position. The lower crosspieces of these sections, as stated, butt against the 1 × 4-in. strip along the top of the side walls. Add the inside sections until the end of the building is reached, when a pair of end sections must be put on. The completed roof may have four cracks between the 2 × 4-in. pieces of its end sections and the top 2 × 4-in. pieces of the gable sections. These may be covered with a board 5 to 8 in. wide, fitted as shown. The whole building is then to be covered with tar felt held on with lath. With the dimensions shown, the floor sections will not fit for a building over four sections long. For a longer building it is advisable to cut off enough on one row of floor sections to prevent their projecting beyond the walls, as a projection tends to allow moisture to run in on the floor. The cut sections can be put next the door, which is ordinarily in the end of the building. They receive there the most wear and being cheap can be discarded as worn out when the building is moved.

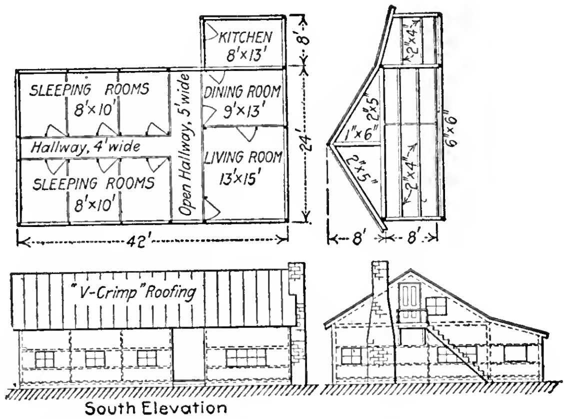
On each erecting sheet is given a list of supplies and equipment as shown in the table, with blank spaces where the number or quantity can be filled in. When the size of building has been decided upon, the bill of materials is made out exactly and the possibility of a shortage of material is thus minimized.

Table of Specifications

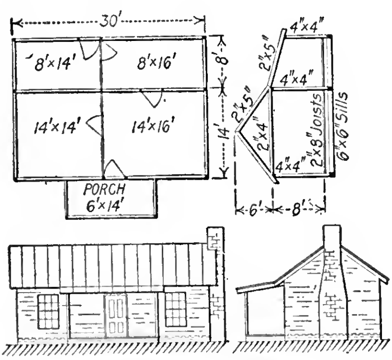
Windows—12 × 24-in. (2 lights)	No.
Stove-pipe saddles (22 × 24-in. rise 4 in 8)	No.
Bolts, ½ × 4½-in	No.
½ × 5½-in. (usually 8)	No.
½ × 6½-in. (usually 8)	No.
Hinges	No. pr.
Door latches	No.
Nails, 40-penny	Lb.
10-penny	Lb.
shingle	Lb.
Tar felt	Rolls.
Lath	Bundles.
Boards—1 × 4 in	Lin. ft.
Boards, extra	
2 × 6-in. by 10-ft	No.
2 × 4-in	Lin. ft.

Inexpensive Miners' Dwellings (By P. E. Barbour).—One boarding house, two four-room houses and three two-room houses, in accord-

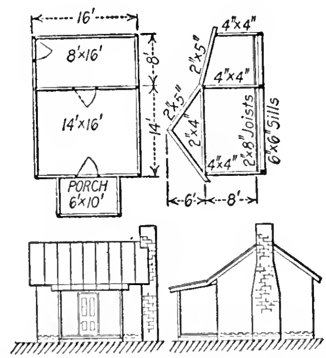
ance with the illustrations of Fig. 2, were built complete at the Montgomery mine, Candor, N. C., for $1223. The buildings were of plain boards, battened up and down, and the rooms inside were finished overhead only. The sills were 6 × 6 in. and the corner posts 4 × 4 in.



Boarding house—Sills, 6 × 6 in.; corner posts, 4 × 4 in.; studding, 2 × 4 in.; girts, 2 × 4 in.; floor joists, 2 × 8 in.; sheathing, 1-in. boards, vertical, with 3-in. battens. Roofing 28-gage V-crimp. Ceiled inside overhead only, walls left rough. Sash made to slide, no weights or cords used.



Four-room house—Sash, 6-light, 10 × 12 in. Doors, 2 ft. 6 in. by 6 ft. 6 in. Chimney, 4 ft. Roofing, 28-gage V-crimp. Shiplap or German siding. Interior ceiled overhead only. Inside walls left rough.



Two-room house—Sash, 4 singles. Doors, 2 ft. 6 in. by 6 ft. 6 in. Chimney, 4 ft. Roofing, 28-gage V-crimp, Siding, shiplap or rough boards with battens. Ceiled overhead only.

FIG. 2.—TYPES OF MINE DWELLINGS AT THE MONTGOMERY MINE, CANDOR, N. C.

All roofs were covered with 28-gage, V-crimped roofing and painted. Lumber cost $12 to $13 per M and labor, from $1.25 to $3 per day. The two-room houses were for negro laborers; the other dwellings were for whites. Rent at the rate of 50 cts. per room per month was charged for all company dwellings, being $1 for the negroes and $2 for the

whites per month. These dwellings were good enough for the climate and were considered desirable by the dwellers.

Ventilating Bunkhouses (By George S. Rollin).—A device in logging camps for maintaining ventilation in the bunkhouses is equally applicable to mining bunkhouses. The stove, Fig. 3, is usually set near the center of the building; for the device in question, a secondary stove pipe is erected as close as possible to that of the stove, beginning 8 to 12 in. above the floor and extending through the roof; it is provided with a damper. In one corner of the building, a hole is cut through the wall, a short length of stove pipe inserted, and connected with an ell to a downright section. This latter also extends to a point 8 to 12

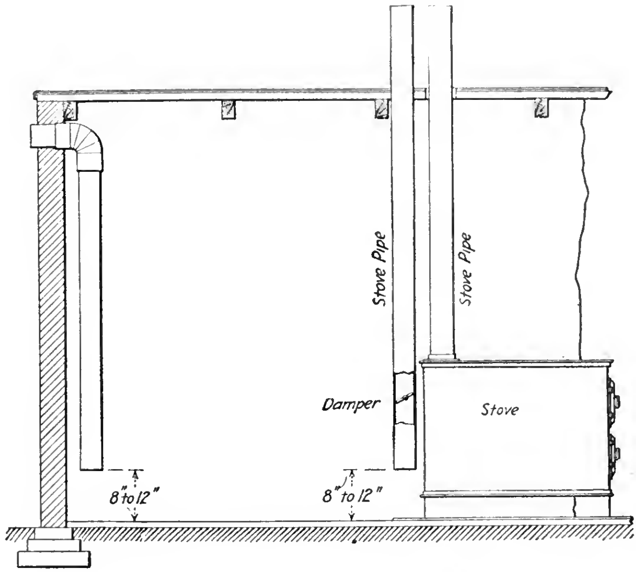


FIG. 3.—ARRANGEMENT OF AUXILIARY VENTILATING PIPES.

in. from the floor. The pipe next the stove is heated and tends to draw up the air, thus adding to the amount of ventilation. The pipe from the outside carries the air to a point near the floor, so that direct draft is avoided, and the incoming air is also somewhat warmed before discharging into the room.

POWER PLANT

Trench and Pipe Feed-water Heater.—A simple, yet successful method of utilizing the exhaust steam from hoisting engines to heat feed water, is shown in the accompanying drawing, Fig. 4. It was designed for a Mesabi mine, where before its introduction the steam from the hoist, as is usual in many plants, exhausted into the air through

a small boiler used as a preheater, a method that proved most unsatisfactory.

A trench was dug from the shaft to the boiler house, a distance of 140 ft., open to the atmosphere at the shaft end through a Y. The ditch was lined with concrete and covered with 3-in. planking, 28-gage corrugated iron and 1 ft. of dirt. The exhaust steam from the hoist discharges into the conduit at the engine house against atmospheric pressure and passes through it into the air at the other end. The water from the mine in a 3-in. pipe passes from a sump at the shaft through

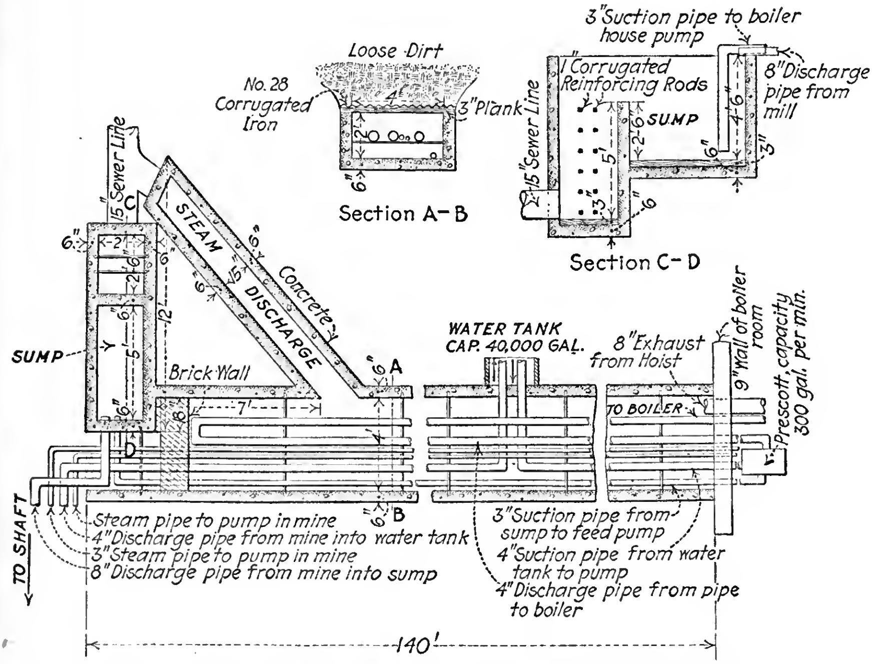


FIG. 4.—WATER AND STEAM PIPES IN CONCRETE-LINED TRENCH.

the conduit to a 300-gal. per minute Prescott feed-water pump in the boiler house. It is picked up by this pump and forced through a 4-in. pipe to the shaft and back into the boiler. The temperature of the water entering the boiler is about 180° F. in contrast with 110° when the old method was used. In case of accident to the underground pumps, water can be drawn from the storage tank through the connections shown. The conduit also carries a 2-in. and a 3-in. steam pipe to the mine pumps. The company using this method figures that it has saved about 10 per cent. of its coal consumption without adding any upkeep cost.

Timber Foundations for Engines (By A. Livingstone Oke).—Sometimes, for reasons of economy or for temporary purposes, it is necessary to use timber for the foundations of small engines, such as hoists, pumps, etc., and Fig. 5 shows one method which gives a rigid setting. In the example shown, several feet of soft soil overlies a rock bottom; in such cases it is desirable to make the foundation of such depth that it will rest directly on rock, as shown. An alternative is to place long, heavy bearers transverse to the upper framing and bolt to these, but the former method is always to be preferred. It is particularly necessary to place the bolts inclined as shown, for the reason that end movement of the foundation tends to tighten an inclined bolt, while the converse is true with a vertical one. The bolts may be made jagged at their bottoms and run in with neat cement, sand and cement, or sulphur.

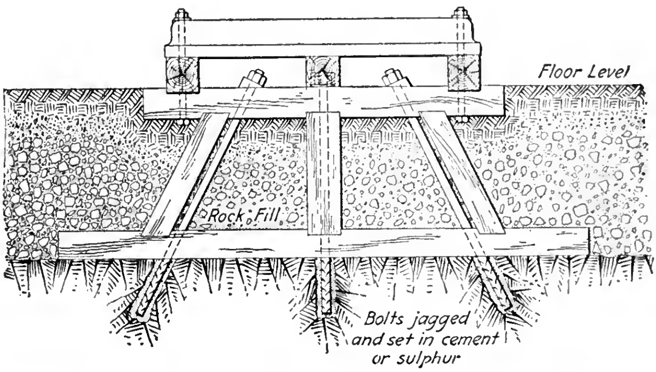


FIG. 5.—TIMBER FOUNDATION FOR ENGINE ANCHORED TO ROCK.

Foundations for Prospecting Machinery (By Horace F. Lunt).—Timber foundations are perfectly satisfactory if properly designed, and if they are not intended for use for a longer time than the life of the timbers; this period will, of course, vary with the kind of timber and the conditions under which it is used. The illustration, Fig. 6, shows a timber foundation designed for a 6 × 8-in. steam hoist. Its essential feature is the platform between the sills, on which sufficient rock and earth is piled to hold the hoist steady. A foundation of similar design but of heavier (12 × 12-in.) timbers for a 30-hp. electric hoist has been used with perfectly satisfactory results. This design has also been employed, using round timbers and round poles in place of the plank platform, the upper set of timbers being hewed flat to receive the bedplate of the hoist. Round timbers make it a little more difficult to level the foundation, but are otherwise just as good.

In building such a foundation the following notes will be found useful:

The holes for the anchor bolts should be bored ¾ in. larger than the bolts, in order to allow for the variations of the bedplate castings. The holes should be bored after the timbers are framed and placed in position; otherwise there is likely to be difficulty in getting the bolts through them. The lower ends of the anchor bolts should rest on something solid, a good-sized flat rock or a substantial piece of timber, so that they cannot be driven down out of reach by any accidental blow before the hoist is in position. The upper ends of these bolts should project far enough through the holes in the bedplate to permit of their being grasped with a pipe wrench in case they turn when the nuts are being tightened. If possible, the excavation for the foundation should be large enough for a

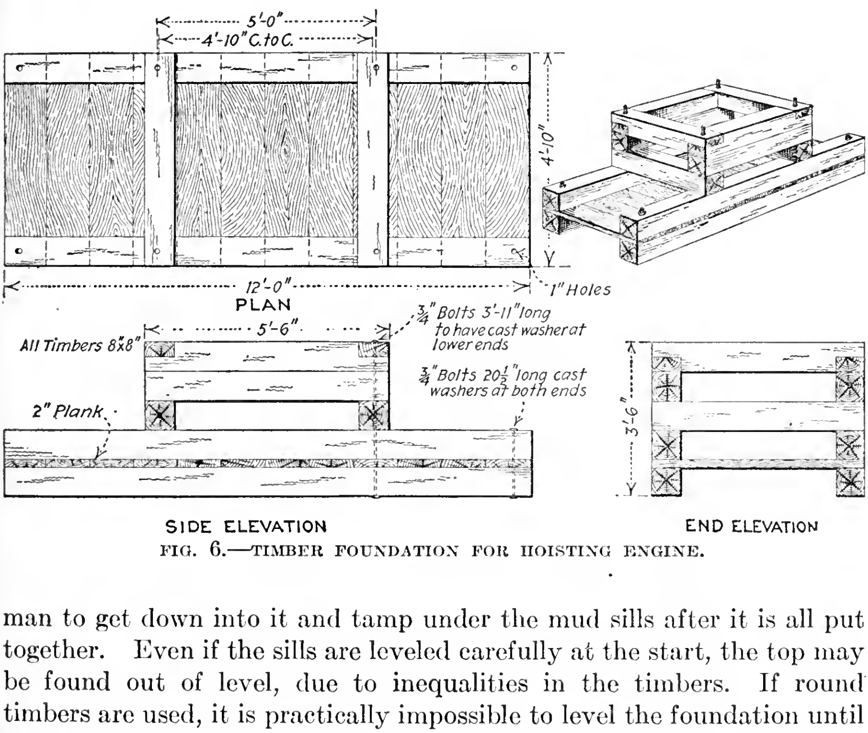


FIG. 6.—TIMBER FOUNDATION FOR HOISTING ENGINE.

man to get down into it and tamp under the mud sills after it is all put together. Even if the sills are leveled carefully at the start, the top may be found out of level, due to inequalities in the timbers. If round timbers are used, it is practically impossible to level the foundation until it is completely assembled. Setting the foundation with its greatest dimension parallel to the line from the hoist to the shaft gives the maximum stability. I have, however, placed the sills at right angles to this line, where circumstances prevented their being laid the other way, making them somewhat longer in proportion to the rest of the foundation, so as to increase the area of the platform and, therefore, the weight holding the foundation in place. This design can easily be adapted to small compressors or other machines. Where a belted compressor is

used, a satisfactory construction is to use the same sills as a base for both compressor and motor foundations.

Automatic Belt Tightener (By J. R. McFarland).—An unusual automatic belt tightener providing a new use for worn-out stoping drills was devised by C. C. Griggs, superintendent of the Uncle Sam Mining Co., at Eureka, Utah. In changing a straight-line steam-driven air compressor to motor-power belt drive, the pulley of the compressor was placed on the shaft between the two narrow-rimmed fly wheels. On account of the proximity of the cylinder, the pulley had to be smaller than good practice would allow. The necessity of having the belt tight on high pressures was apparent. A wooden frame was erected about midway of the belt. On each of the perpendicular posts a feed cylinder from a worn-out stoper was placed. To the bottom of the feed pistons a

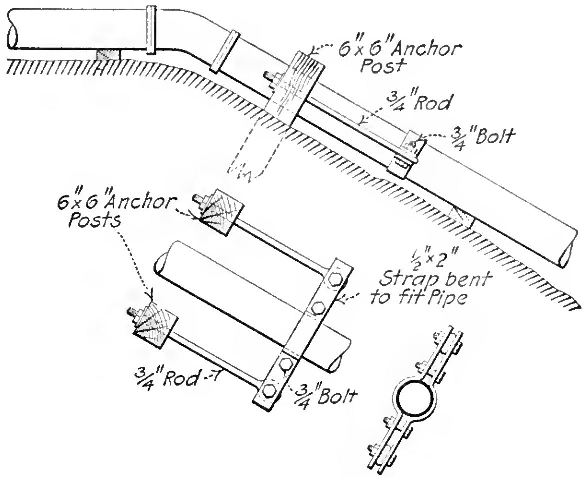


FIG. 7.—ANCHOR POSTS, BOLTS AND YOKE.

shaft bearing an idler was attached with the idler bearing directly on the belt. The tops of the feed pistons were connected with the air line to the receiver. As the air pressure rises in the receiver and the strain becomes greater on the belt with an increased tendency for belt slippage, the air pressure working on the stoper feed pistons holds the idler tightly against the belt and as the air pressure in the receiver drops the tension on the belt is automatically released.

Hillside Pipe-line Anchor (By W. R. Hodge).—To take up some of the weight of a pipe line laid on the surface of the ground and pitching steeply, the arrangement shown here, Fig. 7, is used. Two 6 × 6-in. posts, sunk well in the ground, hold each a $\frac{3}{4}$-in. rod. These rods have at the end an eye through which they are bolted to a yoke. This yoke encircles the pipe to be supported. The yoke is made in two parts and

is drawn up snugly about the pipe by means of two ¾-in. bolts set close to the pipe.

Cheap Expansion Joint (By Fred D. Smith).—For installing a 3-in. steam line 800 ft. long, to furnish a temporary supply of steam from a distant boiler to the hoisting works at the Snow Creek mine, the usual expensive expansion joints were dispensed with and in their stead "shears" were used, made of the same size pipe as the main line, and so called because as the line expands, the mains slide past each other and the upright nipples *A* become crossed, thus appearing as shears.

The accompanying drawing, Fig. 8, shows the arrangement of the device, which requires five nipples and six ells. The nipples *B* and *C* may be of any length, however short, while the nipples *A* should be of a length proportionate to the length of the steam line. In this 800-ft. pipe line the nipples *A* were made about 30 in. long. No strain or

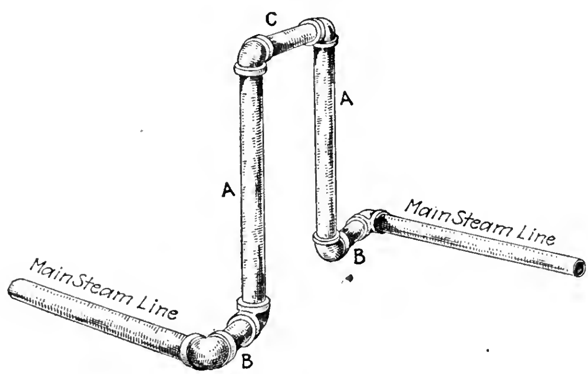


FIG. 8.—EXPANSION JOINT OF NIPPLES AND ELLS.

tension can come on the ells since the nipples can all turn in the elbows. It is suggested that the pipes at these joints should be put together with graphite to insure easy movements of the nipples in the elbows. Since there are no less than six threads on which these turns can be accomplished, there is little or no danger of a sufficient turn in any one elbow to open the joint and cause leakage. As but one of these arrangements was used in the 800-ft. line, the movement in the shears amounted to about 3 ft. While this device in a steam line in a shaft has not been used so far as known, yet it should be available if one were set about every 100 ft., thus requiring probably only about a 12-in. length in the nipple *A*.

SHOP APPLIANCES

Variable-stroke Air Hammer (By J. R. McFarland).—At the Giroux Consolidated Mines Co., at Kimberly, Nev., an air hammer for the

blacksmith forge was constructed out of an old 3¼-in. rock drill. This is usual practice but the delicate control of which the hammer is capable warrants its description. The arrangement is patterned after the design commonly used on drill-steel sharpeners and works much more satisfactorily than the customary throttle valve to govern the operation of the hammer. With a throttle valve the length of strokes cannot be varied and when the throttle is closed the hammer and piston drop to the bottom and are in the way. With the improved arrangement, shown in Fig. 9, both the power and length of the stroke are governed by a small rod extending through the front end of the valve chest, hindering or releasing the valve. The rod has a small shoulder on the inside to prevent its coming out of the chest. The rod on the outside end fits against a lever with a spring attached to one end, holding back the valve when at rest and allowing air to enter in front of the cylinder

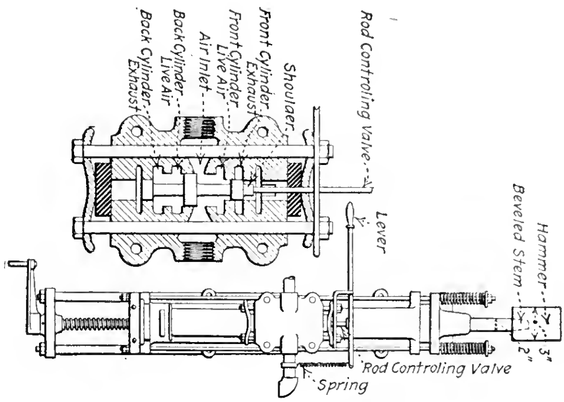


FIG. 9.—ARRANGEMENT OF HAMMER AND VALVE.

and hold the piston up out of the way. As the handle of the lever is pulled back slowly, the air is first allowed to enter the back end of the cylinder in small quantities so that the strokes are light and short, the piston being reversed immediately after the reverse port is passed. As the lever is pulled farther back the drill gets its full supply of air and the momentum of the piston gives it a longer stroke. As the rebound from an iron anvil is greater than from rock a rubber buffer is placed in the back head. The neck of the piston was originally 1½ in. but as a 3-in. square hammer is used the neck was increased to 2 in. The end of the piston neck or stem has a beveled fit in the hammer. The control is so delicate as to allow a piece of paper to be picked off the anvil by the hammer without the hammer's touching the anvil.

Stand for the Blacksmith Shop.—A handy little device in use in the blacksmith shop of the Sterling Iron & Ry. Co. at Lakeville, N. Y.,

is shown herewith, Fig. 10. It serves as a stand for resting the free end of long drill steel in process of being sharpened or for other long work on the anvil. The supporting base is a discarded handwheel such as is used for rack-and-pinion chute gates or for car brakes. Into the square center hole is fastened a short piece of 1¼-in. pipe. In this is telescoped a piece of ¾-in. round iron; this is riveted through a horizontal piece of iron bar with its ends turned up, on which the work can rest. The height of this T-shaped standard can be adjusted in the pipe stand by means of a nail thrust through a hole in the pipe and one of several holes in the rod, the latter spaced about 3 in.

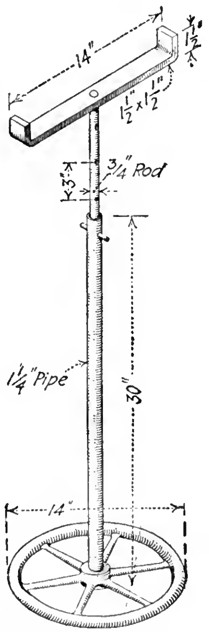


FIG. 10.—HOMEMADE SHARPENING REST FOR LONG STEEL.

Some Smithy Appliances (By A. Livingstone Oke).—The accompanying drawings, Figs. 11, 12, and 13, illustrate a few appliances in the smith's shop at a mine in British Columbia. A circular forge made of light sheet iron is shown in Fig. 11; it may be 4 or 5 ft. in diameter. Air is supplied by a large pipe a few inches lower than the top of the sheet iron; there are three vents shown but only one is usually required, except for big heats. A sliding rest is shown at the back, consisting of a single piece of iron, bent to a U-shape, the two longer limbs passing into the side of the hearth and underneath the air pipe. At the side is shown a swinging arm which is also curved horizontally to lie close around the outside of the hearth when not in use. It is

worth noting that whenever possible the air pipe at the side should be bent over and taken underneath the floor level, as this leaves the forge clear all around for working.

A small furnace for heating drill steel, which is inserted from both

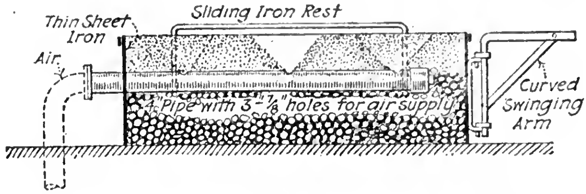


FIG. 11.—CIRCULAR FORGE WITH RESTS ATTACHED.

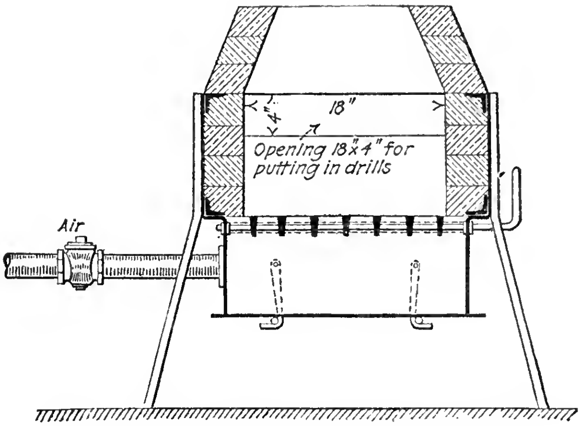


FIG. 12.—FURNACE FOR HEATING DRILL STEEL.

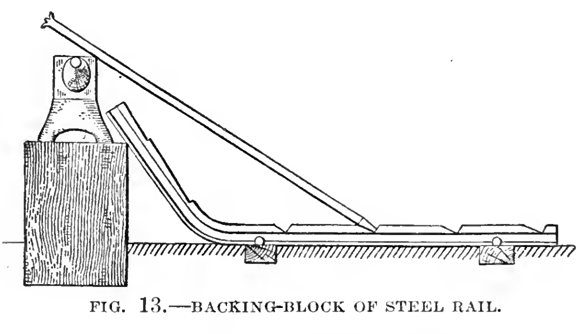


FIG. 13.—BACKING-BLOCK OF STEEL RAIL.

sides in the openings near the top, is shown in Fig. 12. The body of the furnace consists of sheet iron riveted to two rectangular frames, at top and bottom. Inside this body a firebrick lining is laid and carefully fitted. The bottom of the furnace proper has a revolving grate, worked by a handle at the side. Below the grate is the air-box, which also

receives the ashes and from which they are discharged through the hinged door, held up by the links as shown. There is a slightly conical cupola-top, through which the coke may be added.

A simple footing or backing block for holding drill steel up to the anvil when swaging is shown in Fig. 13. It consists of a piece of 20- or 30-lb. rail, bent as shown and laid on two or more sleepers, with suitable notches cut in it to receive the ends of the drill steel. The top of the rail should be spiked to the timber under the anvil, as well, to assist in holding it solidly up to the work.

Emergency Backing-block.—In Fig. 14 is shown a backing-block made of a rail in somewhat the same manner as that just described but with the position of the rail reversed. Except for being somewhat light, it answered the purpose as well as a standard cast block. A piece of 60-lb. rail, 3½ ft. long, was bent as shown, and holes were cut in the

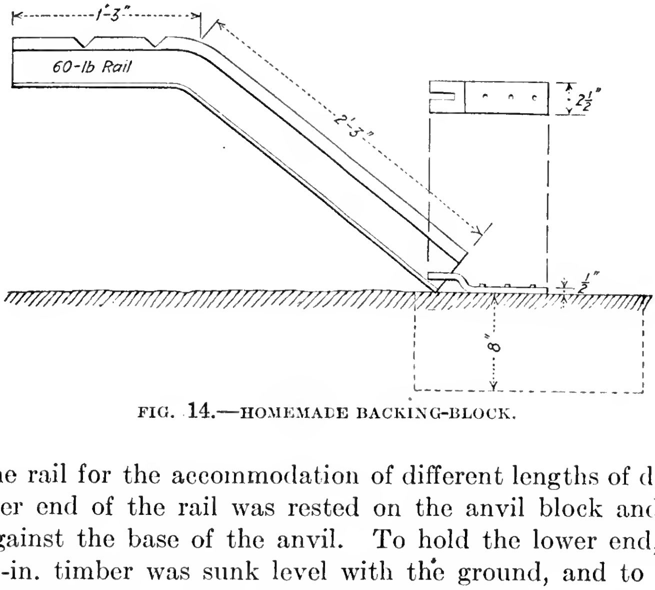


FIG. 14.—HOMEMADE BACKING-BLOCK.

top of the rail for the accommodation of different lengths of drill steel. The upper end of the rail was rested on the anvil block and pushed firmly against the base of the anvil. To hold the lower end, a piece of 8 × 8-in. timber was sunk level with the ground, and to this was firmly spiked a 10- or 12-in. piece of ½ × 2½-in. iron; one end was bent up as shown, and had a slot cut in it just to fit the web of the rail.

Coke Furnace for Heating Drills (By Claude T. Rice).—The coke furnaces used at the Champion mine, at Painesdale, Mich., for heating drills for sharpening and tempering have made the low record for coke consumption of less than half a pound of coke per drill sharpened and tempered. The reason for this low consumption is that the coke, as well as the air, is preheated, while the drills are heated for tempering on the same fire as those being heated for the machine sharpener.

The furnace, as shown in Fig. 15, is made with a double firebox, each half being about 24 × 9 in. inside the brick, and these fireboxes are separated from one another by a partition made of firebrick piled lengthwise across the wall. The firebox is 10 in. deep and fitted with a shaking grate designed somewhat on the principle of the grates in the firebox of a locomotive. From time to time the grates are shaken and the clinkers broken up so that there is practically no burning out of the grate bars.

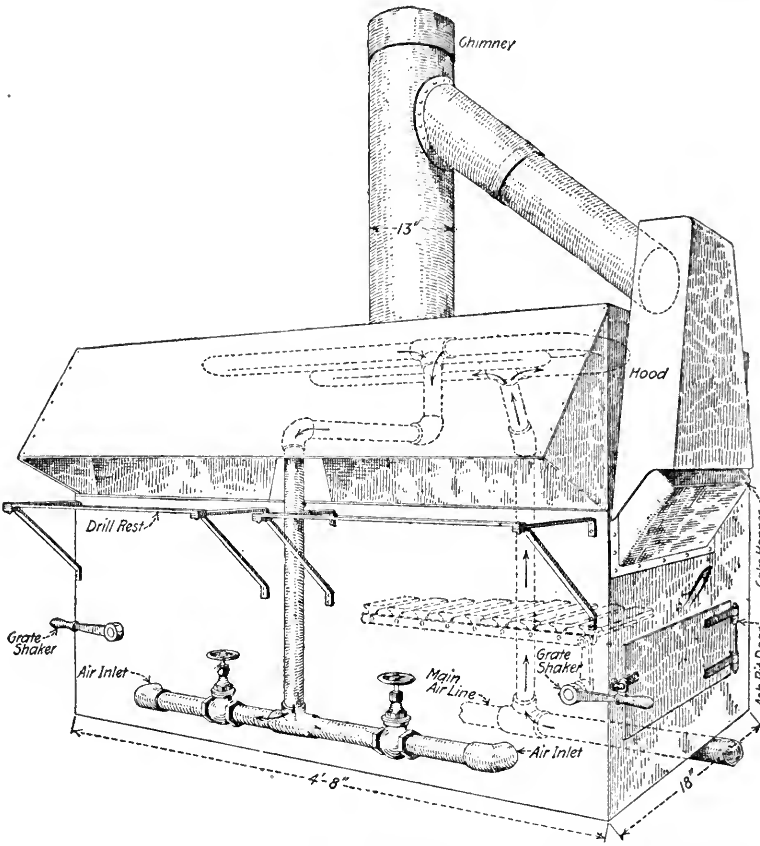


FIG. 15.—DRILL-HEATING FURNACE USED AT CHAMPION MINE.

The wear on the bricks of the furnace is such that the furnace has to be relined only once in two weeks, 12 firebricks being required for the purpose. This is practically the only maintenance charge there is. At each end a hopper projects from the body of the furnace. These hold the coke and are kept filled so that the coke is thoroughly heated before going down on the fire. This coke feeds down into the firebox as the fire needs it, but, from time to time, the fire has to be poked, and the coke

spread out. This is done through the coke hopper, a hood being used to take out of the shop through the main chimney of the furnace any gases that may come up the hopper. This hood is carried by a sleeve that goes over the main chimney pipe so that it can be made to serve the other hopper by swinging it around, holes being cut in the main pipe to permit this. Generally, only one end of the furnace is used at a time, and the heating of the drills for sharpening is done on one side, while on the other side of the fire the drills are being heated for tempering. The path of the air through the furnace is shown. It comes in at the bottom and is taken up through the furnace to a coil of pipe on top of the furnace arch; thus the hot gases have to play around the coil before going out the chimney; the air then comes down to the bottom of the furnace again. In this way any water in the air is converted into steam and the air is preheated. This causes less formation of clinkers than would be the case if wet air were blown under the grate.

Joplin Steel-sharpening Kinks (By Claude T. Rice).—In the lead and zinc districts of southwestern Missouri, "bull" bits, or, as they are

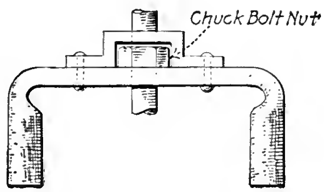


FIG. 16.—HANDLE FOR CHUCK NUT ON SWING PISTON.

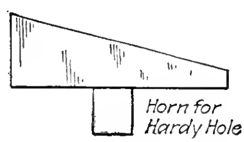


FIG. 17.—BEVEL SWAGE.

more generally called, chisel bits, are employed even on machine steel. It is maintained that these cause less trouble in "blocking the hole," the local term for fitchering. Furthermore, it is possible for one man to sharpen the chisel bits, while two are usually necessary for cross bits. This is a point of great importance in a small property. It is customary to have one man alone do the sharpening for an entire mine up to 125 drills per day.

To handle the larger and heavier pieces of machine steel, however, special devices are necessary. Some of these are illustrated in Figs. 16, 17, 18 and 19. To back up the piece of steel being sharpened an old machine drill piston is used. This is carried in a horizontal position by a suspension hook such as is common for long work on the anvil. The shank end of the drill being sharpened is inserted in the old piston chuck. The piston has one nut of the chuck bolt removed and the other nut fitted with a device for setting it up quickly, Fig. 16.

To hold the bits on the "stoving" block an old machine drill chuck is frequently set in the block under the anvil so that the drill shank

enters it when the drill is laid on the stoving block. This latter is usually a piece of 80-lb. rail with its head buried in the ground, the steel to be stoved resting on the upturned bottom of the flange. The chuck prevents swinging of the steel if it is not struck quite evenly while stoving.

Two anvils are generally used, the second one being under the air hammer at the side of the shop, which is an old piston drill mounted on a pipe sunk in the ground. The process of sharpening is as follows: The hot steel is first placed on the stoving block and stoved back with a two-handed sledge weighing from 8 to 10 lb. After the metal has been stoved

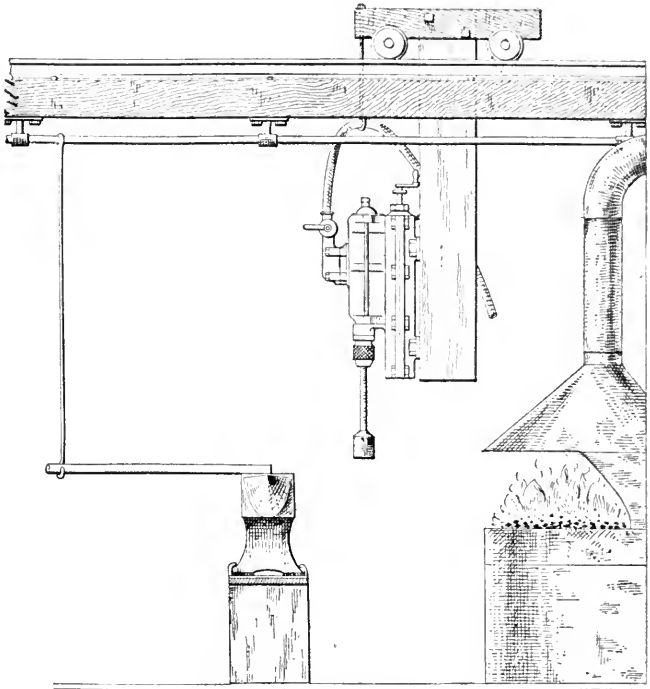


FIG. 18.—SHOP WITH MOVABLE HAMMER.

back to form the bit, which has a face angle of 130° to 140° ordinarily, the steel is transferred to the anvil under the machine and spread out and battered down to gage. The end of the piston in the machine has a slightly rounded face. The long pieces are supported in the swing which runs on a piece of 2-in. pipe across the top of the shop. From this roughing anvil, the steel is transferred to the finishing anvil. If it is a short piece, the piston is found useful to back up the bit with its weight and relieve the smith from shock. The tightening device, shown in detail, is made of a piece of old drill steel fitted to the chuck-bolt nut and is rather heavy so that by slipping the drill shank in the chuck of the

swinging piston and giving the handle a spin, the bolt is sufficiently tightened. The weight of the longer drills is sufficient to take the shock of the dressing blows. In dressing, the bit is usually held up at a small angle so as to bevel down the sides slightly. A hook somewhat higher than the ordinary hook is necessary for this, but the same result can be got by the use of the bevel swage, shown in Fig. 17, which has a horn to fit the hardy hole of the anvil.

The layout of one shop, presented in Fig. 18, is somewhat different. In this shop all the work is done on one anvil and the stoving block and roughing anvil are dispensed with. The machine is mounted on an 8 × 8-in. timber, which runs on an overhead track by means of a truck, as shown, and can thus be brought into service on the one anvil. The air hose for the machine is kept out of the way by suspending it in an eye-bolt hung from the crosspiece of the truck.

A unique method of tempering the steel is used. It is a modification of the plunging method that retains all the advantages of plunging while there seems to be little trouble from temper checking and few

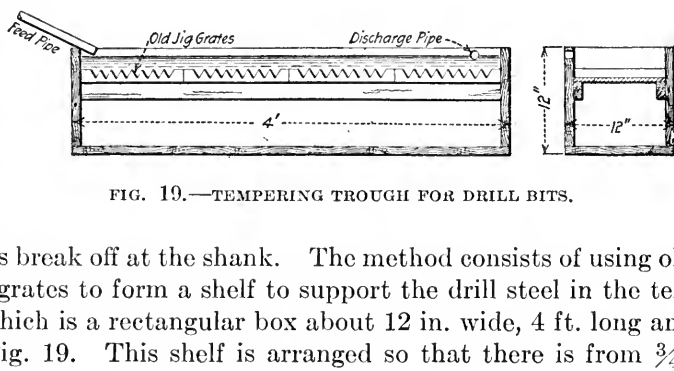


FIG. 19.—TEMPERING TROUGH FOR DRILL BITS.

drill bits break off at the shank. The method consists of using old worn-out jig grates to form a shelf to support the drill steel in the tempering tank, which is a rectangular box about 12 in. wide, 4 ft. long and 12 in. deep, Fig. 19. This shelf is arranged so that there is from 3/4 to 1 1/4 in. of water on top of the grates. In a few instances more than this depth of water is used over the grates, but that is not good practice as it is likely to result in breaking off of the steel at the water line. The shelf must be made of iron, for if wood were used, the hot drills would burn into it enough so that the bit would not be properly cooled, and a soft drill would result. A greater depth of water is used over the grates with machine sharpening than with hand work, owing to the fact that it is more difficult to keep the water cool with the faster sharpening done with machines, than when the work is done by hand. There is a continuous feed of water going to the tank, and the amount is regulated so that when a drill has been put into the tank the temperature of the water is approximately at the temperature of the inflowing water. By this method the tempering can be done in one heat just as in barrel plunging and there is no necessity of bathing the steel up and down in

the water to prevent its sudden cooling at the water line and the breaking that would result. The reason for this is that the drill bit is covered with so shallow a depth of water that the steam generated by the hot steel causes the water line to play up and down on the metal and does the bathing automatically. The accompanying drawing shows the trough used in this method of tempering steel. All the tool sharpener does is to hold his steel for the right temperature, which is approximately that at which he finishes the sharpening operation, and to stand the drill on the grate in the tempering box. Once he has the water regulated properly, he has to pay no more attention to it. Both bull bits and cross bits are tempered thus. Some blacksmiths in the district say that they have been able to get as good results by this method as by draw tempering. Draw tempering is generally used on the hand steel, although some hand steel is also tempered by this plunging method. The machine steel chiefly used is of the lower carbon grades not containing more than 0.60 per cent. carbon.

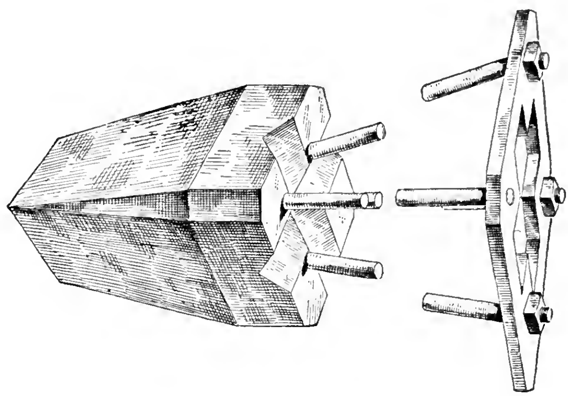


FIG. 20.—SWAGE FOR SHARPENING DRILLS BY HAND.

Sharpening Machine-steel by Hand (By Emory M. Marshall).—For sharpening machine-steel at a mine where the sharpening is done by hand, the device shown in Fig. 20, designed by S. J. Collins and used by the McConnell Mines Co., is of value. The ordinary dolly used for sharpening is drilled at the four corners and rods of 7/16-in. iron are screwed into each hole. The dolly is then mounted in an anvil block or fastened securely to the floor. The rods at the corners may be of any length, say 12 to 18 in. The shortest length of steel that can be sharpened depends upon the length of these rods, so they must be short enough to accommodate the starters. These rods are fastened at the top about 2 in. apart by bolting to a plate of 1/8-in. iron, which has been cut out at the center to allow the drill to pass through easily. The

method of sharpening is the same as the old method, but instead of holding a dolly on the bit while a helper strikes it with a heavy hammer, the blacksmith drops the heated steel between the rods and pounds it up and down on the dolly. The iron rods are set so close to the corners that the steel cannot turn, while the flare at the top allows the steel to work freely. Of course, the blacksmith cannot strike as heavy blows as his helper can with a hammer, but by working the drill steel up and down 12 or 15 times, he can get as good, or a better, bit than he can by having a helper to strike for him, and he can do it in the same or less time. This method enables one man to sharpen drill steel as fast as two by the ordinary method of hand sharpening, and the miners claim that the bits are better.

Tempering Hand-drill Steel (By Albert G. Wolf).—A simple but satisfactory method of tempering hand-drill steel was originated by John Sommers, foreman of the Blue Jay mine of the Mason Valley Mines Co., and is in use at that mine. It consists of heating the drill to a dull red for a distance of about 1 in. back from the edge of the bit, then

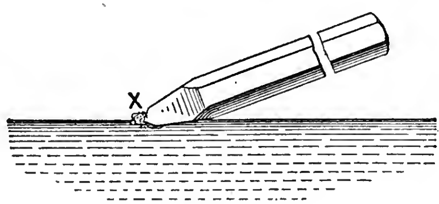


FIG. 21.—CONVENIENT METHOD OF HAND-STEEL TEMPERING.

placing one side of the bit just flush with the surface of the water, as shown in Fig. 21. The water will climb up on the bit at X as long as sufficient heat remains in the steel. When the water ceases to climb, the steel is removed and allowed to cool slowly. At the time of removal the temper is extremely hard, but the heat remaining in the steel will draw back into the bit and toughen it. Drills sharpened in this manner never check, as the first cooling is almost uniform throughout; corners of the bit are rarely broken off; and the edge of the bit is resistant to the wear against hard rock. The temper can be regulated closely by the degree of water cooling.

U-Bolt Bending Tool (By Dan Fields).—In Fig. 22 is shown a device for bending piston-machine U-bolts without battering the threads. It is adapted for making bolts for a 3⅛-in. drill, although it can be used for smaller bolts if they are given one blow on the anvil. To use the machine, the bolt is heated, placed on the center, clamped with the handle and bent down with a wood or rawhide mallet. The square-section projection on the bottom should be of the proper size to fit the

anvil hardyhole. The handle is about 3 ft. long and made of 1-in. round mild steel. It gives sufficient leverage to hold the bolt firmly.

Compressed-air Jet for Cleaning.—Where many machine drills are being operated, much time may be saved to the repair man by using an air jet for cleaning various drill parts, such as cylinders, barrels, front heads and valve chests. A ½-in. pipe line is connected to the receiver or air main, wherever it is most convenient, and a line is run to the work bench. A globe valve is placed in the line, and beyond the valve, two ¼-in. lines are taken off. One consists of a nipple, followed by a globe valve and

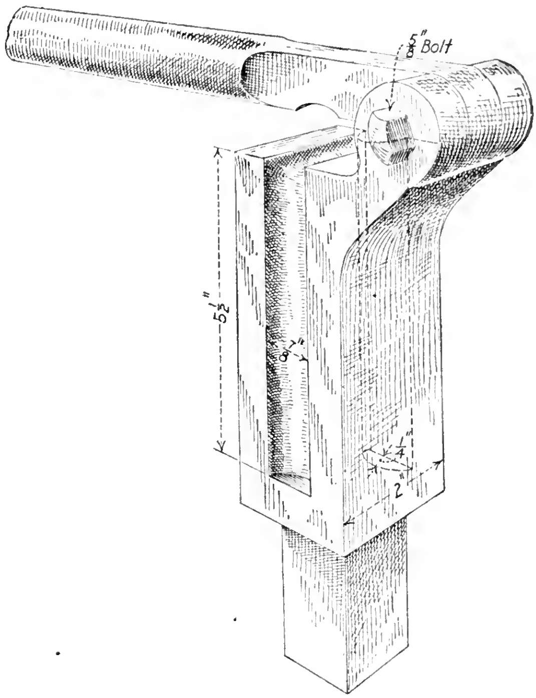


FIG. 22.—DEVICE FOR SHAPING MACHINE-DRILL U-BOLTS.

a 1-ft. length of ¼-in. pipe. The end of this piece is drawn out to form a nozzle. The other consists of a globe valve followed by a length of ½-in. rubber hose and a ¼-in. nozzle. The first jet is convenient for cleaning small parts which are easily handled; the second for larger parts, or parts which may be clamped in the vise.

Devices for Bending Plates (By Claude T. Rice).—It often becomes necessary to bend iron plates in a mine shop, particularly where the mine manufactures its own cars. In the shops of the Desloge Consolidated Lead Co. of southeastern Missouri, two devices are in use for this purpose.

In Fig. 23 (elevation) is shown a vise, made of two 60-lb. rails set

horizontally, one below the other, with the flanges facing each other. The plate is inserted between these rails, which are screwed together and the bending done by hammering. The lower rail is set on two posts in notches, and projects a few inches beyond each post. The web and head are cut away at the ends and the flange drilled to permit the passage of the vise screws. These screws, made from the feed screws of machine drills, work in nuts, which are under the flange, and have lugs bearing

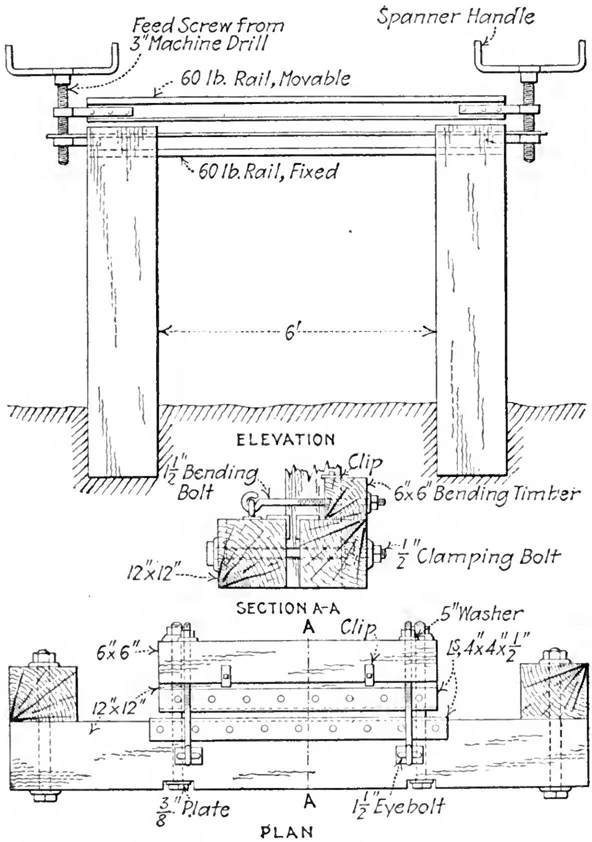


FIG. 23.—CLAMP AND BENDING DEVICE FOR STEEL PLATES.

against the posts to prevent turning. Collars are riveted to the web of the upper or movable rail which is shorter than the lower rail and these collars project sufficiently to allow the screws to work through them. In the drawing the upper rail is shown lifted to admit a plate. The tops of the screws are fitted with double-ended spanners to serve as handles for tightening and loosening. The vise is capable of handling plates up to ⅜ in. in thickness.

An apparatus which includes a bending device and eliminates hammering is shown in Section A-A and in plan, Fig. 23. It consists of two timbers side by side, by which the plate is clamped vertically, and another timber moving on bolts, which does the actual bending. The clamping timbers are 12 × 12 in., their upper adjacent edges being bound with 4 × 4 × ½-in. angles. Two 1½-in. horizontal bolts with suitable washer plates and nuts on one end, are used for clamping. The opening between the timbers is adjusted by these to any width, so as to control the sharpness and the amount of the bend. Through the fixed 12 × 12-in. timber are two vertical 1½-in. eye-bolts and linked into these so as to form a hinge are two horizontal 1½-in. eye-bolts. These latter pass through the 6 × 6-in. bending timber, and by means of nuts on their ends, this timber is drawn horizontally against the vertical clamped plate and bends it down. On the top bearing edge of this timber are two clips or buttons which can be turned out to come over the top of the plate and by swinging the timber up somewhat, the bending of the plate can be started above the clamping line.

Homemade Timber-framing Plant (By Frank M. Leland).—In order to avoid purchasing an expensive timber-framing machine, the Empire Copper Co., of Mackay, Idaho, devised a rig out of scrap machinery from about the mine, together with a new slabber including saw, and a swinging cut-off saw. Fig. 24 shows the general features of construction. A frame was built of 8 × 8-in. timbers and the cut-off saw frame laid horizontally upon it instead of vertically. An old vertical 7 × 10-in. engine was connected as shown, using channel irons with slotted holes so that the saw frame could move up and down, cutting any depth desired, or could be lowered clear down and used to square the timbers. Another saw running horizontally was rigged up. This does not cut at the same time as the vertical saw; hence, only power enough is required to drive one saw at a time. The guide rails under the table were made from 1½-in. square iron set edgeways; the rollers were turned out of some matte pans; the pulleys and shafting were picked up around the place. The machine is strictly homemade, but it is good and strong and it runs perfectly.

The posts are first sawed on the slabber; they are next squared on the ends by running through the machine, say one hundred of them; the saw is then raised up so as to cut just 2 in. deep. The horizontal saw remains stationary; the stick is shoved through and the cut made on top; the carriage is then shoved on the horizontal saw and a slice taken out of the bottom; it is then pulled back and rolled over one-quarter and the operation repeated; it requires four cuts to finish the end. It averages 3½ min. to frame both ends of an 8 × 8-in. post.

The slabber takes logs up to 8 ft. in length. A set of dogs was made

and some perforated plates to hold pins, and an inserted tooth saw used to replace the thin saw which came with the slabber. The machine will not only make square-set timbers but, using logs 8 ft. long, it will make lumber for just about one-half its cost when purchased from the saw mills. It will also saw 2 × 4-in. stuff for ladders, 4 × 6-in. for flooring for the square sets and almost anything up to 8 ft. long. The 2 × 12-in. by 8-ft. lumber is found just as good for chutes, etc., as is 16-ft. The ends are squared before sending to the mine, making the planks exactly 8 ft. long and by putting the sets in the chutes on 4-ft. centers, there is no sawing to be done in the mine by candle light with a dull saw; this saves money. A little wedge saw was rigged to use up the scraps, and wedges cost 3/8 ct., as against 6 cts., made by hand. Ladder rounds 2½ in. square by 16 in. long are made by ripping them on the small mill. The great advantage from the whole plant is that from every timber are got four slabs, two of which are generally good for lagging and

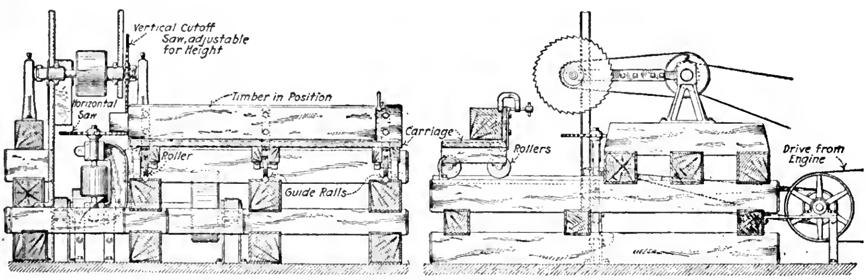


FIG. 24.—FRAMER, SHOWING BED, CARRIAGE AND SAWS.

in almost every case enough lagging is had from the timber to pay for the timber itself, since 2-in. plank would be used otherwise. Furthermore, the boiler, being purposely left naked, warms up one end of the shop and for the other live steam in coils of pipe is used; the fuel costs nothing since sawdust is burned. This is quite an item, as it ordinarily takes from 300 to 400 lb. of coal daily to keep the shop warm.

HANDLING SUPPLIES

Pipe Rack.—A satisfactory and convenient pipe rack for the mine warehouse is illustrated in Fig. 25. The larger pipes are stored on the base and lower arms, and lighter pieces on the upper arms. Iron bars and structural shapes may also, of course, be handled in the rack. The rack possesses the great convenience of permitting the pieces to be extracted either lengthwise or sideways, and by setting it in the middle of the room, both sides are made accessible.

It consists of a number of bents spaced 4 ft. apart. Six bents would thus give a length of 20 ft., sufficient to accommodate standard pipe. Five bents would probably be just as satisfactory. Each bent consists of a center post 8 × 8 in., with inclined pieces 6 × 6 in. symmetrically placed on each side. The base is an 8 × 8-in. by 10-ft. piece and suitable bracing is provided at the bottom. The line of bents is tied by a

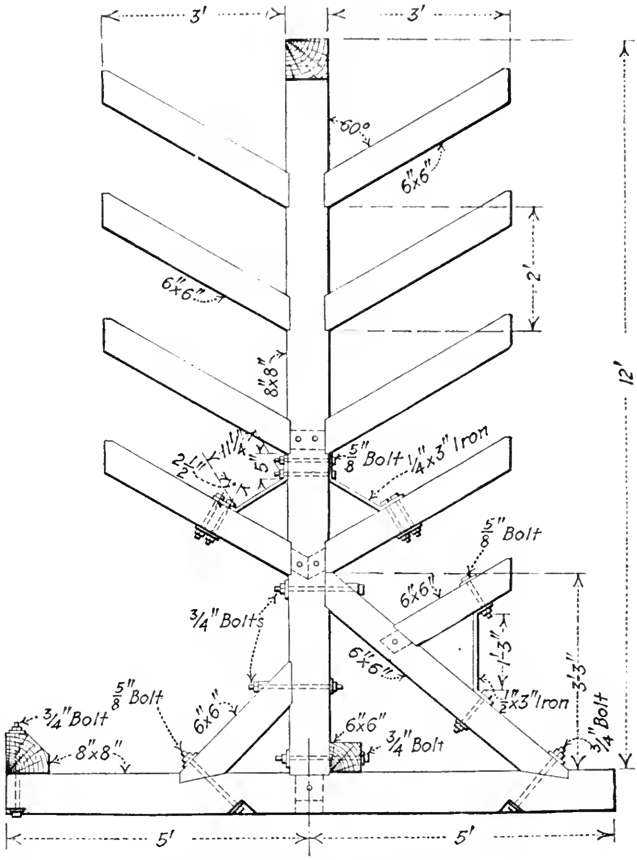


FIG. 25.—END VIEW OF PIPE RACK.

horizontal 8 × 8-in. piece along the top of the center posts, a 6 × 6-in. piece at its bottom and an 8 × 8-in. chamfered piece along the ends of the bases. The upper diagonal arms should be supported by iron braces similarly to the lower set, although these are not shown in the illustration. These 6 × 6-in. diagonal arms should also be mortised into the center post with either inclined or square mortises, as shown on the lower two sets.

Carbide Container and Measurer (By E. W. R. Butcher).—With the introduction of carbide lamps, it has been found unsatisfactory to allow the miners to help themselves from the carbide cans. The Republic Iron & Steel Co. therefore furnishes each of its miners with a small screw-top can which holds enough carbide to last one shift. These cans are filled from the carbide container, a drawing of which is shown

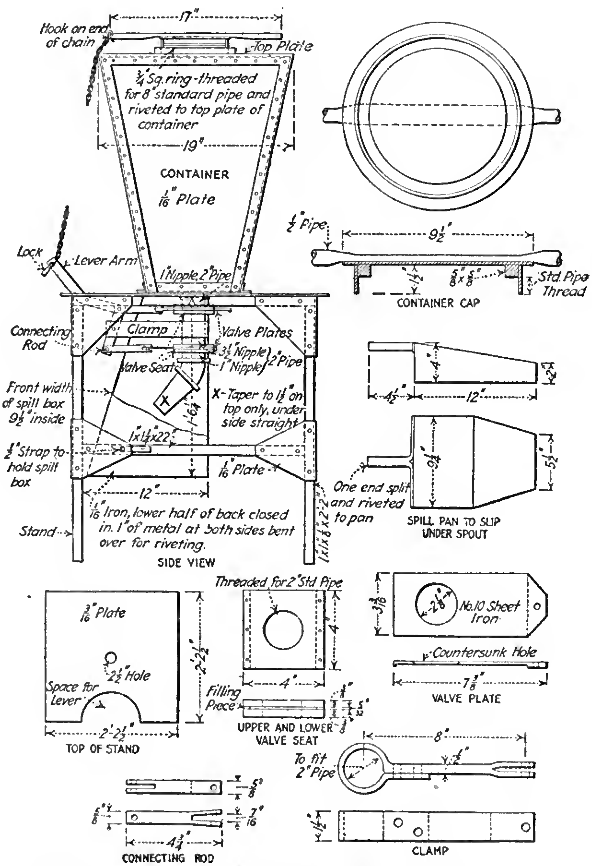


FIG. 26.—ASSEMBLED CARBIDE CONTAINER AND DETAILS OF CERTAIN PARTS.

in Fig. 26. By means of the valves, one motion of the lever arm, up or down, gives out just enough carbide to fill a miner's can. The spill pan is used to catch any carbide which falls. Since the container has been in use, the carbide consumption has been reduced nearly one-half.

A screw top with a ½-in. pipe for a lever handle is provided on the top of the container. The carbide issues from the container bottom through a 2-in. pipe connection. In this connection are set two valves,

each consisting of a seat and a plate, in both of which are $2\frac{1}{8}$-in. holes. The plate slides over the seat, and when the holes register, the valve is open and carbide can pass. When the holes do not overlap at all, the valve is closed. The sliding back and forth of the valve plates is effected by the lever arranged to close one valve while opening the other. Between the valves is a $3\frac{1}{2}$-in. nipple of 2-in. pipe, which forms the measuring box. A chain connection between the pipe handle of the cap and the end of the operating lever can be kept locked. The details of the construction of the entire apparatus can be seen in the drawing.

STEAM-SHOVEL WORK

Changing Dipper Teeth on Steam Shovels (By Clarence M. Haight). —When digging iron ore with a steam shovel, it becomes necessary, at intervals of two or three weeks, to replace the dull dipper teeth with sharp ones. As there are four teeth used on a dipper, the time thus

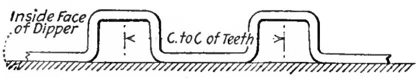


FIG. 27.—HOLDING STRAP.

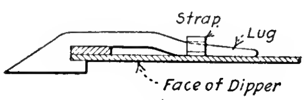


FIG. 29.—TOOTH INSERTED.

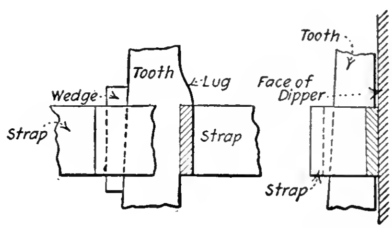


FIG. 28.—TOOTH IN STRAP

spent in the course of a season is an important item. Formerly a tooth was fastened to the dipper with six 1-in. bolts. With this method much trouble was experienced; for, unless there was absolutely no play between the dipper and the tooth, the whole strain of the digging came on the bolts, which bent so much that it was difficult to remove them. On this account the changing of a tooth was generally done on Sunday in order not to delay operations. As it is not necessary to change all the teeth at the same time, a part of almost every Sunday had to be devoted to this work.

At many mines now a new method of attachment is in use, which makes the removal and the replacement of a tooth a comparatively simple matter. A strap of $\frac{3}{4} \times 5$-in. iron, bent as shown in Fig. 27, is riveted inside the dipper on the face which digs the ore. The dipper tooth has a lug projecting from one side near the back end, as shown in Fig. 28. The loops in the strap are wide enough to admit the tooth and this lug easily. When the lug is behind the strap the inset part of the tooth fits against the edge of the dipper. The end of the tooth is

pushed to one side until it touches the side of the strap. A wooden wedge is then driven in between the tooth and the other side of the strap to hold the lug in place, completing the operation. The lug prevents the tooth from falling out and the edge of the dipper takes all the strain, as it should. The inserted tooth is shown in Fig. 29. To remove the tooth the operation is reversed; the wedge is driven out, any dirt in the way is removed, the tooth is shoved to one side until the lug will clear the strap, and the tooth is then pulled off.

Grade Board (By L. B. Pringle).—A grade board used extensively in steam-shovel mining around Hibbing, Minn. is shown in Fig. 30. The novel feature of the device consists in the fact that the measurements are so adjusted that the terms "per cent." and "inches" are interchangeable.

The necessary equipment consists of a board, 1 × 8 in. by 10 ft., having its two edges carefully planed and parallel, of one small carpenter's spirit level, of two round iron pins ¾ × 24 in., having standard flat bolt heads, and of a supply of wood suitable to cut into small blocks

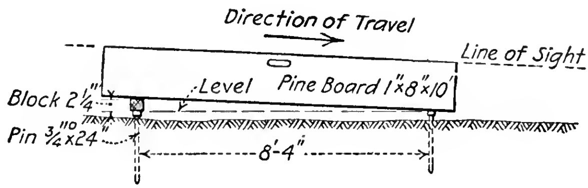


FIG. 30.—GRADE BOARD IN USE.

of various thicknesses. A handle 1 × 1½ × 5 in. may be cut in the board to facilitate carrying. The pins are spaced exactly 8 ft. 4 in. apart, parallel to the direction of travel of the shovel, and are driven close to the ground. By means of the board and level the pins are brought to the same elevation. Then for down-grades a grade block is placed under the board on top of the rear pin; for up-grades, under the board on top of the front pin. For a 1.0 per cent. grade there is used a 1-in. block; for a 1.5 per cent. grade a 1½-in. block; etc. The illustration shows the application of the method in running a 2.25 per cent. down-grade. The shovel runner sights along the upper edge of the board at a graduated stick or rule held upright by a helper at various points along the line, and knowing the depth below grade he is to carry throughout the course, and the height of the top of the board above the ground, he can readily ascertain the depth of cut needed at any point. The percentage of error increases with the length of sight taken from a set-up; but by making two and sometimes three set-ups a day the error found by checking with the engineers' leveling is so small that it can be disregarded.

Track Connections.—Much time is lost in steam-shovel work in "moving up." This loss is usually due either to derailment on account of poor track laying or to the slowness with which the crew makes the connections. In order to save time the arrangement for connecting track sections, illustrated in Fig. 31, has been devised and used in many pits on the Mesabi iron range. Not only does it save time in laying the track, but insures a connection that the wheels will pass over without danger of derailment. It consists of two 4 × 6-in. angles, shop riveted to a ¾-in. plate, which is as wide as the tie used and long enough to accommodate the size rail used in the sections. The angles are spaced ¼ in. wider than the width of the base of the rail. The holes for connecting the rails are spaced and bored as in ordinary angle bars, except that there is but

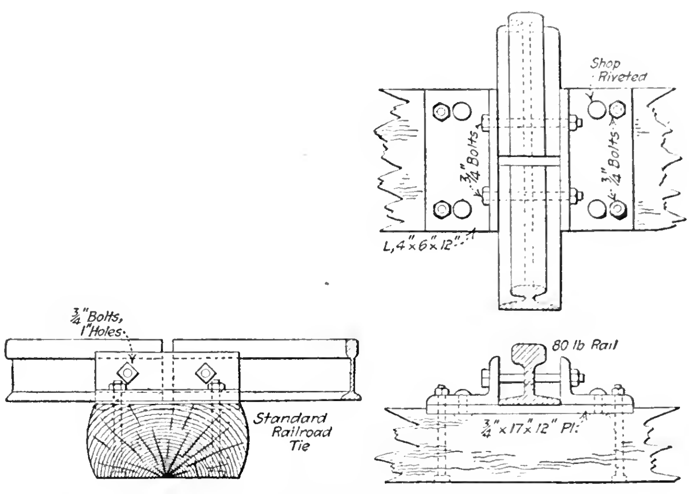


FIG. 31.—ANGLE-BAR, STEAM-SHOVEL TRACK CONNECTIONS.

one hole for each rail, which is ¼ in. larger in diameter than the bolts in order to lessen the difficulty in making the joint. Two plates, with their upright angles, are bolted to a good tie, as shown in the accompanying drawing. It is customary to bolt the connecting tie to the front section while the steam shovel is at work. As soon as the whistle blows to move up, the rear section is carried forward; and it is only necessary to pass a bolt through each plate to make the connection instead of having trouble with two pairs of angle bars.

Rigid Track-connection (By E. C. Hingston).—The rail chair or track connection illustrated in Fig. 32 has the advantage over that just described of affording greater rigidity, as it fits over the base of the rail and prevents a sideways roll. The chair, made by the Bucyrus Co., is a steel casting fitting the rail loosely, its length corresponding to the width

of a tie, 8 in. In steam-shovel work, the new track usually lies on an up-grade, since the shovel tends to imbed the section on which it rests. This fact, combined with the rough bottom and the necessity of providing for side swing, accounts for the large clearance allowed, the short length of the casting and the placing of the holes for the pins near the ends of the rolls. The pins are not threaded and can be quickly inserted and removed. They are chained to the casting to prevent losing. Both of the rail ends lie on one metal plate and the tops are thus always maintained at the same level, which is not always possible when using bridle-bars. More clearance is allowed on the sides of the rail base than shows in the illustration, in order to permit swinging the rail in making a curve.

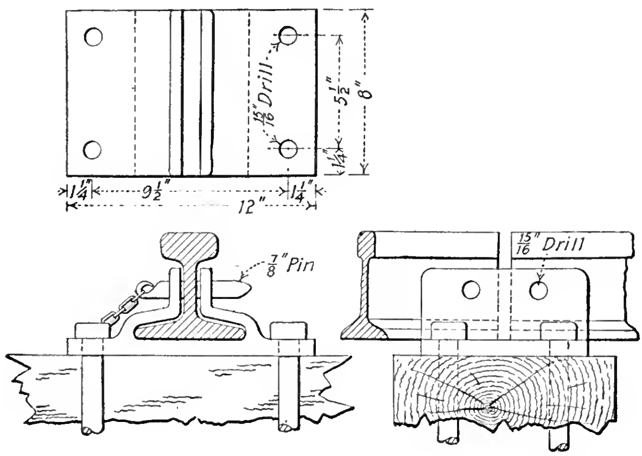


FIG. 32.—CAST-STEEL TRACK CONNECTIONS.

Backing Block (By L. B. Pringle).—The block-and-clamp device of Fig. 33, was developed and adopted for its steam shovels by the Oliver Iron Mining Co., Hibbing, Minn. It is of simple construction, is quickly and cheaply made, is easy to handle, and has proved a success. The block is placed against the hind wheel of a steam shovel and prevents any backward movement of the machine as the dipper gouges into the dirt, or when the shovel is working on an up-grade.

The material for block, clamp and pin is wrought iron. The block is made from 1 × 2¼-in. stock bent into a right-angled triangle with dimensions as shown. The clamp is of 1 × 2½-in. stock, made to fit snug over the rail and through the block. The holes in the sides of the clamp are cut to fit a pin 1 × 1¼ in. tapered to 1 × 1 in. and 10 in. long. The distance from the top of the clamp to the upper edge of the holes is 7½ in., thus allowing ½-in. play beneath the bottom of the rail. There is approximately 10 in. of longitudinal play between the clamp and the inner edges of the block. This permits the clamp to be placed against a

tie, and the block so adjusted as to have its working face over the tie or beyond it. If the block is inclined to slip, a small iron or wooden wedge inserted between the block and rail at the rear end will make it secure. Unless the shovel is unusually large or is working in extremely hard

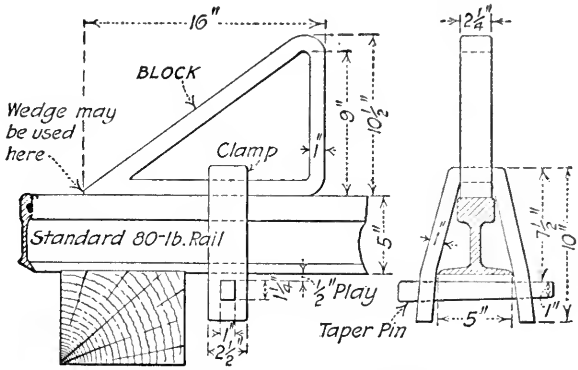


FIG. 33.—ASSEMBLED PARTS OF STEAM-SHOVEL BLOCK.

ground or is on a maximum up-grade, one block and clamp will be sufficient.

MISCELLANEOUS NOTES

Cooler for Drinking Water (By E. W. Durfee).—For those living in arid regions, where a supply of ice is not always available, the apparatus

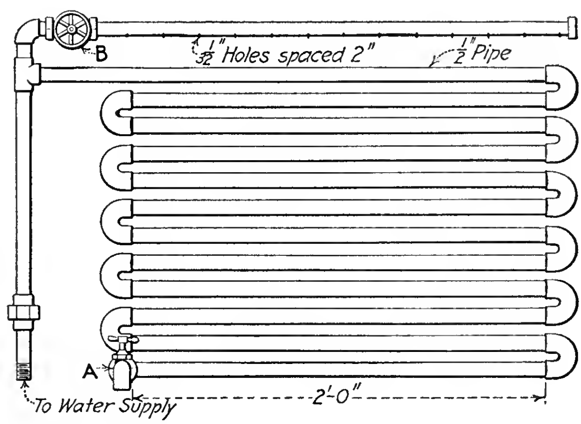


FIG. 34.—SMALL-SIZE RADIATOR-TYPE WATER COOLER.

herein described will be found satisfactory for supplying cool drinking water. One has been in use at the Alvarado mine for at least two summers and has been found superior in every way to the *olla*, so commonly used throughout the southwestern United States. It is more efficient as a

cooler, is perfectly sanitary and does not require the attention for filling and cleansing that is necessary with the other device. Its efficiency, of course, depends upon the amount of humidity in the atmosphere, but for average conditions in Arizona the difference between the temperatures of the wet and dry thermometers in summer is about 30°, and sometimes runs as high as 40°. Fig. 34 shows the construction, the whole being made up of ½-in. pipe and fittings. The pipes and return bends should be wrapped with thin cloth and placed in a shaded open place, where the prevailing winds will cause the maximum evaporation. The drip from the upper pipe should be regulated by means of the valve *B* to just the amount necessary to keep wet the pipes below. As the cool water is drawn from valve *A* it can readily be seen from the drawing that the pipes are kept filled from the water supply. With an apparatus of the size shown, about 10 glasses of cool water can be drawn at one time, after which it will take about 15 min. to cool another supply to the minimum temperature it will attain.

[An excellent portable alternative device is a coarse-canvas bag with a bottle neck sewed into one corner, the "South African Water Bag."—EDITOR.]

Elevating Guy Lines (By L. E. Ives).—The height at which guy lines generally cross paths is just about that necessary to strike one in the neck. It is possible to overcome this difficulty at slight expense and at the same time provide a perfectly strong anchor. A concrete post, reinforced with two 30-lb. rails, is made with a total length of 10 ft. The upper 6-ft. section of this post is circular in cross-section and 12 in. in diameter. The remainder is square, 20 in. on a side. The square section is placed in the ground, and the round portion extends above ground. An eye-bolt is embedded in the top, and the guy line is attached to this. In this way the line at its lowest point is 6 ft. above the ground, giving clearance usually sufficient.

Arc-light Tower.—The details of construction of an arc-light tower, made chiefly of pipe, and used at the mines of the St. Louis Smelting & Refining Co. in southeastern Missouri, are shown in Fig. 35. These towers are used to support the lights that illuminate the yards in the immediate vicinity of the main shafts and shops.

The legs *A* of the towers are pieces of 1¼-in. pipe set in concrete pedestals at a batter of ½ in. per foot. The three pipes are tied together at intervals of 4 ft. by means of the crossbraces *B*, made of ⅜ × 1¼-in. steel cut to the proper length, together with the standard clamps *J*, made of the same size of steel, bent to the shape shown and held together by bolts, 20° malleable-iron angle washers *E* being used on the coupling bolt *F*. At the top there are two triangular plates *C* and *D*, that slip over the legs. These are made so as to ride the legs about 18

in. apart. The caps I are screwed on the top of the legs after the plate has been put on. In the center of these plates a hole is bored to receive the pipe G, also made of two pieces of $1\frac{1}{4}$-in. pipe put together with an elbow and reinforced by an angle brace of $\frac{3}{8} \times 1\frac{1}{4}$-in. steel.

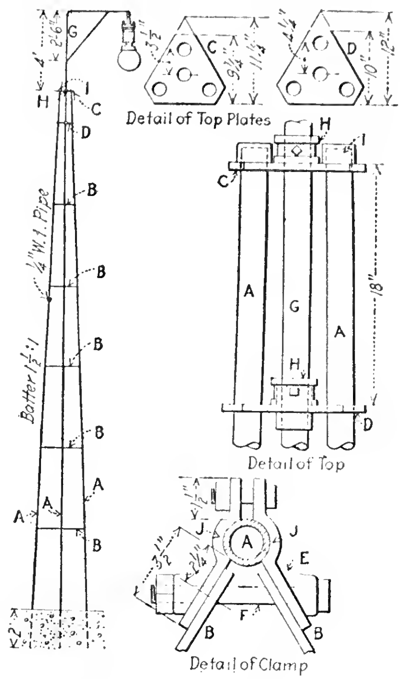


FIG. 35.—ARC-LIGHT TOWER FOR SURFACE USE.

On this top piece, which carries the arc light, are two collars H, that rest upon the two triangular plates. Owing to the way that the three legs are strapped together, all the fittings are standard and the cost of the towers is small.

II

EXPLOSIVES

Blasting and Handling Explosives—Storage and Thawing—Safety Precautions

BLASTING AND HANDLING EXPLOSIVES

Mammoth Blasting by Electricity (By E. Hibbert).—At the Mother Lode mine of the British Columbia Copper Co. in British Columbia, the ore is broken off in large slices at infrequent intervals, several hundred thousand tons being broken in one blast. Electrical blasting is used for this work. In the big blast, of October, 1911, the method used was similar to subsequent blasts, and may be described as typical.

Loading started on the morning of Oct. 2, at 7 a.m., and continued until the morning of Oct. 4, at 1 a.m. All the machinemen were employed loading the holes under the supervision of the foreman and shift bosses. The detonators were given to specially selected men, whose sole duty during the loading was to make up the primers and hand them to the loaders, seeing that the wires of the detonators were carefully unwound and free from kinks. The electrician commenced stringing lead wires on the morning of Oct. 3, and when this was finished he started wiring up the most distant sections, connecting all holes in series of 25 to the lead wires. At 1 a.m., Oct. 4, a special wiring crew, consisting of the superintendent, foreman, shift bosses and a few picked men was put on under the charge of the electrician and the wiring completed. The blast was exploded at 10:30 a.m. and broke 175,000 tons of ore.

In this blast 2433 holes were loaded with 10⅝ tons of 40 per cent. dynamite. The holes were connected to the lead wires in series of 25, using 2525 Nobel's No. 7 low-tension detonators with 8-ft. lead wires, the extra detonators being used to make up the series of 25 detonators in a circuit in places where the holes could not easily be connected up to the lead wires in series of 25. When everything was ready for the blast, the main lead wires were connected to a 550-volt, alternating-current circuit. The main lead wires from the transformers were of No. 1, rubber-covered copper wire, and from these No. 6 and No. 8 weather-proofed copper wires led off. No. 10 and No. 12 weather-proofed copper wires again led from the No. 6 and No. 8 wires for the shorter circuits. All joints on the leads were covered with friction tape.

The No. 7 low-tension detonators with 8-ft. lead wires have a resistance of one ohm and require ½ amp. for detonation. In the blast described, there were 101 circuits with 25 holes in series in each circuit, and as a current of ½ amp. was required in each circuit, a current of 101 × 0.5, or 50.5 amp., was required in the main lead wires, figuring no losses. Had connection been made to the 110-volt lighting system, the total resistance of the leads and detonators would require to be under 110 ÷ 50.5, or 2.2 ohms. With the wiring used this did not leave enough margin for safety, figuring on poor joints, etc., and so connection was made to the 550-volt circuit, especially as no instance had been noted at this mine where excess of power caused any trouble with complete detonation.

In connection with electrical blasting, care must be taken that the resistances in the various circuits are well balanced, so that all the detonators will explode at the same time. The best way to insure this is to use large lead wires and to take care that the same number of detonators are connected up in each series. This was illustrated in a rather forcible manner. On one occasion a small blast had to be exploded and the electrician did the wiring for it. He had been instructed to see that all detonators were connected up in even series, but on this occasion there were 101 holes to be connected, and he took the chance of making three series of 25 holes and one series of 26 holes, the last series being nearest to the main leads. After the blast, it was noticed that the three series of 25 holes had exploded, but the series of 26 holes had failed to explode; fortunately the unexploded holes were easy to rewire. It was considered possible that if the speed at which the fuse wires in the detonators reach the detonating temperature was less than the speed of explosion, then, given suitable conditions, the explosion from the series of 25 could break the lead wires of the series of 26 before the fuse wires in these detonators had reached the detonating temperature, and so cause the failure to explode.

Some experiments were made to test out this idea. Two detonators were wired in series and one lead wire was wound round another single detonator. The two detonators were connected through one circuit to the battery and the single detonator through another parallel circuit. When fired, the single detonator exploded, broke the lead wire around it, and the other two detonators failed to explode. This showed that with the current employed the speed of explosion of a detonator was greater than the speed of heating up the fuse wire in a detonator. Similar results were obtained by connecting up in this manner series of two and three, and series of three and four detonators. It is possible that the failure of the 26 holes to explode may have been due to some other cause, but the experiment showed clearly the necessity for having

well-balanced circuits with the same number of detonators in series in each circuit.

Sinking with Delay-Action Fuses (By W. V. DeCamp).—The somewhat recent development of the electric detonator known as the "delay-action fuse," placed on the market some years ago by the Giant Powder Co. and later by the Du Pont company, has neither the disadvantages of the instantaneous electric detonator nor those of the long black-powder fuse, subject to so many causes of failure. The delay-action fuse, Fig. 36, is in reality an electrical means of spitting holes; for it consists simply of a small piece of gutta-percha fuse, on one end of which is placed an ordinary detonator, and on the other a cap containing the igniting material. Through this cap, terminal wires are inserted and connected by a ribbon or sponge of low-fusion material. This sponge is ignited by the passing current and in turn ignites the fuse, which burns to the cap

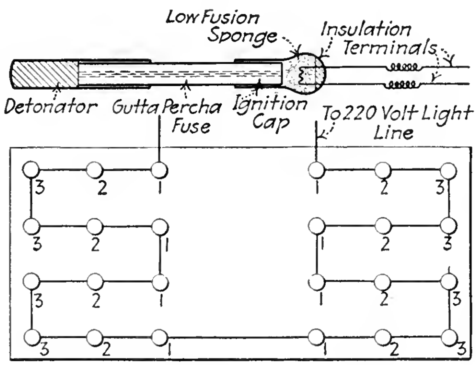


FIG. 36.—DELAY-ACTION DETONATOR AND SHAFT-WIRING DIAGRAM.

and explodes the charge. It is obvious that any length of delay can be obtained by lengthening the piece of fuse. The entire device, fuse, detonator and ignition cap, is incased in a rubber tube and coated with a good waterproofing paint. The terminal wires, where they enter the ignition cap, are also incased in sulphur, insuring perfect insulation. In most underground work only three, or possibly four, delays are necessary. Each fuse with terminal wires attached has a tag fastened to it, stating its number, so that there need be no confusion whatever in loading holes.

A thorough trial of this type of fuse was made at the property of the Pacific Copper Mining Co., Crown King, Ariz., and it resulted in a material reduction in sinking costs. The shaft in which the trial was made was broken about 7 × 10 ft., and while sinking through an extremely hard siliceous schist a large volume of water was encountered; this, with the small dimensions of the shaft and the hard rock, greatly

delayed the work; 20 to 24 holes were required to break a 4-ft. round; missed holes were a regular occurrence and the cost became almost prohibitive. The average time required for a full round was from 36 to 48 hr. and the average depth broken was 4 ft. During the month previous to the use of delay fuses the daily average advance was 2.1 ft. The delay fuses were used on the last 48 ft. of shaft sunk, 10 rounds being required in this distance. Every round fired was satisfactory, there being only two missed holes in the entire 10 rounds.

Comparative Costs of Sinking Wet Shaft with and without Delay-action Fuses

	Ordinary method; depth sunk, 63 ft.; time, 30 days			Using delay-action fuses; depth sunk, 48 ft.; time, 15 days		
Labor:	Total cost	Cost per ft.		Total cost	Cost per ft.	
Miners	$1210.75	$19.20		$617.50	$12.87	
Topmen	242.00	3.84		116.00	2.42	
Engineers	522.80	8.35		310.00	6.45	
Foreman	186.00	2.95		99.00	2.06	
Blacksmith and helper	239.30	3.81		122.00	2.54	
			$38.15			$26.34
Power supplies:						
Fuel oil	648.00	10.30		402.00	8.38	
Lubricants	37.87	0.60		21.00	0.43	
Coal	35.50	0.56		16.20	0.33	
			11.46			9.14
Supplies:						
Machine repairs	22.80	0.36		19.50	0.40	
Powder	194.00	3.09		91.50	1.90	
Fuse	42.15	0.68		*a*24.00	0.50	
Caps	6.60	0.01		......	....	
Candles	34.75	0.56		*b*12.10	0.25	
			4.70			3.05
Timbering:						
Setting	72.00	1.15		45.00	0.94	
Framing	63.00	1.00		46.50	0.97	
Timber	190.00	3.02		146.50	3.05	
			5.17			4.96
Water:						
Fuel and oil	139.50	2.22		210.50	4.37	
Pumping labor	27.00	0.43		124.00	2.58	
			2.67			6.95
Total			$62.15			$50.44

a—Electric fuses.
b—Lamp.

Operations were as follows: After loading, the holes were connected in series and thence to a 220-volt direct-current lighting line on which there were two switches, one at the nearest level, the other at the collar

of the shaft, both under lock and key, the key in possession of the boss. Three different lengths of fuse were used, one length on the eight cut-holes, the second length on the next row of four holes on each side and the third length on the two rows of end holes, as shown in the diagram, Fig. 36. The length of fuse in each case was 1.5 in., 2 in. and 3 in., respectively. From such a round one would naturally expect to get three distinct reports, but such was never the case. There would be from 8 to 16 reports for the entire round; the reports would be close together and sounded much like the rattle of a bunch of firecrackers. The large number of reports obtained was probably due to the fact that pieces of fuse of the same length would not burn down in exactly the same time, and to the probable difference in resistance of the ignition material in different caps, thereby requiring a slightly longer time for ignition.

In all of the 10 rounds the rock was broken much finer than had been the case with ordinary fuse; this was probably due to the rapid succession of the explosions, the rock mass being maintained thereby in a constant state of rapid vibration, resulting in greater fracturing, as the successive charges would explode. The average depth of round using delay fuses was 5 ft., the average depth broken, 4.8 ft., and the average time for each round, 36 hr.

The accompanying table shows the difference in cost of sinking by the two methods. It will be seen from total costs, that there was a large increase in the expense of handling water, also an increase in cost of engineers, due to some heavy repairs made; otherwise the costs check up fairly well.

Blasting Box for Sinking (*Proceedings*, Lake Superior Mining Institute).—A device which is ingenious and, so far as known, unique, was employed in sinking the Palms shaft of the Newport Mining Co. It was a paraffin pasteboard box, 9 × 3½ × 1½ in. deep. In the sides of this box, near the bottom, holes were made with an iron punch just large enough for a fuse to fit snugly. As first used, a positive electric wire was led through one end and a negative through the other, and the ends of these connected with a 1-amp. electric fuse. Two of the boxes were used at once to blast the cut, which consisted of about 40 holes. From the surface, two positive wires of 14-gage copper were strung, one for each box; the negative wires from the boxes were connected to the air pipe. After fuses of proper length were inserted through the holes in the box, a small quantity of F. F. rifle powder mixed with ordinary black blasting powder was strewn over the 1-amp. fuse and the box was covered with a wooden lid. When the men reached the surface they tested with a galvanometer to find out whether the wire connections were satisfactory when the circuit was closed. Then a 250-volt current was thrown on and the 1-amp. fuse was melted so that the powder ignited, and in

turn ignited the fuses leading into the boxes. The cross spitting of the fuses across the boxes made it almost impossible for a misfire to occur. The lengths of the fuses were adjusted to get the proper sequence of holes.

It was found, however, that the preparation of the 1-amp. fuse box required too much labor, so that later an electric blasting squib was used to ignite the powder in the box. A squib was placed through a hole in each end of the box, two being used to insure igniting the powder, and the boxes were connected in series with only one No. 14 positive copper wire from the surface and with a single negative wire connected to the air pipe. Finally, a Du Pont relay electric fuse igniter was used and connected with the squib.

Electric Blasting with Dry Battery (By C. Carleton Semple).—A cheap electric blasting apparatus can be made from dry batteries such as are used for electric call-bell work. Tests show that at least two caps can be detonated for each cell in the battery. This would indicate that a battery of eight cells should suffice for any ordinary drift round. The eight cells may be placed in a wooden box, conveniently in two rows of four cells in the row. The cells should be connected in series, one battery terminal being connected to one of two binding posts placed near together at one end of the box. The other battery terminal should be connected to the center of a two-point switch secured to the end of the box near the two binding posts. One point of the switch should be connected to the second binding post; the second point of the switch, not being connected, is the off position. The box should be equipped with a substantial hinged cover, preferably locked in place, and a stout strap which may be slung over the shoulder for carrying the battery about.

The operation is simple. The switch is carried on the off or blank point. The wires from the round are secured in the binding posts and when ready to blast the switch is slipped over to the other position. The effect is, of course, instantaneous unless an attempt is made to fire too many holes for the number of cells used. While the flow of current is strong, the circuit is broken so quickly by the detonation of the charges that the dry cells should recover quickly and be serviceable for a long time.

It is a simple matter to arrange such a battery so that the circuit may be tested by sending through a small current before switching on the full current of all the cells in series. One pound of No. 22 copper magnet-wire, wound on a spool for convenience, placed in the circuit of only one electric detonator, will enable one to send the current of one dry cell through the circuit without exploding the detonator. If, then, the battery box is so arranged with one cell in circuit with such a spool of wire

for resistance, it will be possible to arrange a three-point switch in such a manner that the lead wires from the round may be attached to the binding posts while the switch is on the blank point. Obtaining a spark at the switch while making and breaking contact with the second point to which the single cell and coil of wire is connected, would indicate a closed circuit which will be fired upon throwing the switch over to the third point to which all the cells of the battery are connected. A still more elaborate device can be made by placing a cheap and simple galvanometer on the end of the box connected with the second point of the three-point switch and the one cell and resistance coil; the movement of the needle would then indicate a closed circuit much better than would sparking at the switch.

Fuse-cutting Table (By Herbert Oates).—To facilitate the rapid cutting of fuse to any desired length a table, illustrated in Fig. 37, was devised. The board *A*, 1½ × 12 in. by 9 ft., is erected to a suitable height

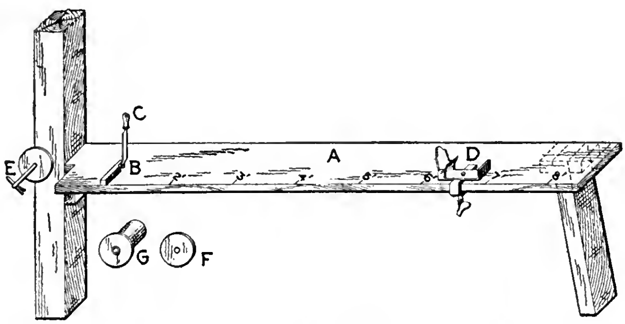


FIG. 37.—FUSE-CUTTING TABLE WITH KNIFE, PINS AND CLIP CLAMP.

and the cutting block *B*, ½ × ½ × 6 in., fastened near one support. The knife blade *C* is hinged to the block at one end, so that its edge will just pass the side of the block and make a clean cut. This edge should be thin and beveled like a wood chisel. The adjustable clamp *D* is fitted with a thumbscrew, so that it can be shifted to give the desired length. The clip on the end has teeth, but is not so rough that the fuse will be cut. The spool *G* is 3½ in. in diameter and 6½ in. long, built about a piece of ½-in. pipe and closed at the end with a disk 7 in. in diameter. There is a ½-in. pin *E*, fastened to an upright and a wooden disk *F*, 1 × 7 in. In operation, the coils are separated into the small inner coils and the larger outside coils, and two of the smaller size slipped over *E* and held on by *F*. The ends are brought to bear against the upright end of the clamp *D*, and the clip forced down upon them. The fuse near the coil is held on the block *B*, and cut with the hinged knife. The outside coils are manipulated by putting them on the spool *G*, two at a time, and then slipping the spool on the pin *E*.

Fuse-cutting Bench (By S. R. Moore).—A good bench used at the Success mine in Idaho is shown in Fig. 38. It is cheap and simple and has proved satisfactory. The bench is 3 × 9 ft. Two rollers, one 2 in. and one 4 in. in diameter, are mounted at one end to hold the fuse coil. One of the bearings for each roller pin is slotted, so that the rollers are easily removable. Across the opposite end of the bench is nailed a 1 × 2-in. slat, and on top of this is a lever made of 1½ × 2-in. material, slightly beveled on its lower edge, with a weight attached to its handle; the fulcrum end is raised just the thickness of a fuse, so that when the weight is applied, the lever bears equally upon all the fuses between it and the slot. A chopping block of 2 × 4-in. material is nailed at such a distance from the lever as to give the desired length of fuse, and in such a manner as to be easily replaced when worn out.

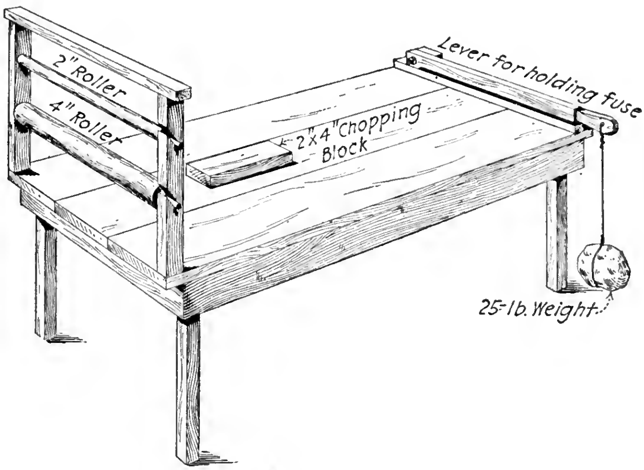


FIG. 38.—BENCH WITH ROLLERS, CHOPPING BLOCK AND WEIGHTED LEVER.

To use the bench, five large coils are placed on the larger roller, and five small coils on the smaller roller. It is found that by piling coils on top of one another, and pushing the roller through before removing the paper, no difficulty is experienced from the coils becoming entangled. The ends of 10 fuses are placed under the lever, and with one blow of a sharp ax on the chopping block, the fuses are easily cut to exactly the same length. The new, evenly cut ends are again placed under the lever before releasing the hold. The cut fuses are rolled in coils of 10 each with the ends to be capped sticking out slightly. If a glancing blow be given on a good chopping block with a sharp ax, the end of the cut fuse will cap better than when cut with a knife and equally as well as when cut with a crimper. With a little practice, 500 fuses can be cut,

coiled and counted in a hour, as compared with half a day when cut singly.

Powder Chutes for Openpits (By B. M. Concklin).—An arrangement has been installed in several of the openpits upon the Mesabi range for the purpose of saving time and facilitating the distribution of black powder to points within the pits. The device, as shown in Fig. 39, is simple in itself, consisting of two greased slides in which run

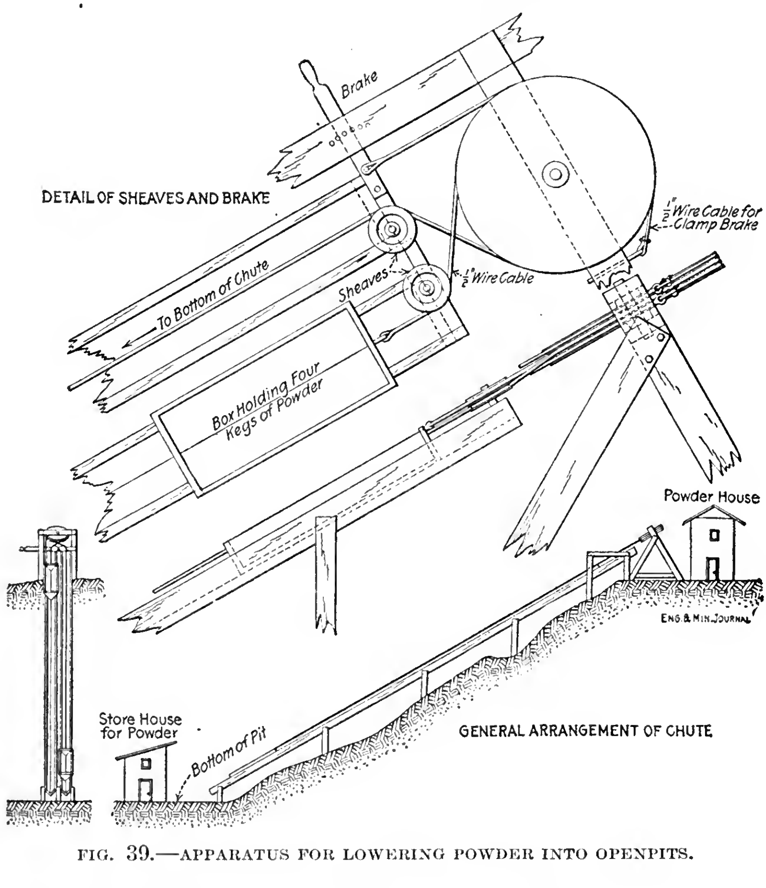


FIG. 39.—APPARATUS FOR LOWERING POWDER INTO OPENPITS.

two triangular-shaped boxes made to carry from four to six kegs of black powder, and an arrangement of sheaves at the top of the slides as shown in the accompanying engraving. The operation is accomplished by means of gravity, the loaded box carrying the empty one to the top. The speed is regulated by means of the brake shown. Where the pits are deep and the lower levels are inaccessible, except by long and steep climbs, this contrivance has been the means of a considerable saving of time and money as well as acting as a safety device

for the prevention of possible explosions when the powder is thrown over the banks into the pits.

Bag for Carrying Dynamite (By W. R. Hodge).—In the Burra Burra mine of the Tennessee Copper Co., it is often necessary to carry a box of powder up or down 150 ft. of ladders to get it into a back stope. The bag illustrated in Fig. 40, has been devised to make easier the transportation of dynamite through the manways. The bag is of canvas, reinforced by a double strip of the same material on the bottom and ends. At the top of each end the strip is doubled over and riveted about an iron ring to which is snapped the shoulder strap. This strap is of leather 2 in. wide and 3 ft. long, provided with a harness snap at each end. The bag is divided by a canvas partition down the middle, making compartments a little longer than a stick of dynamite. A canvas flap

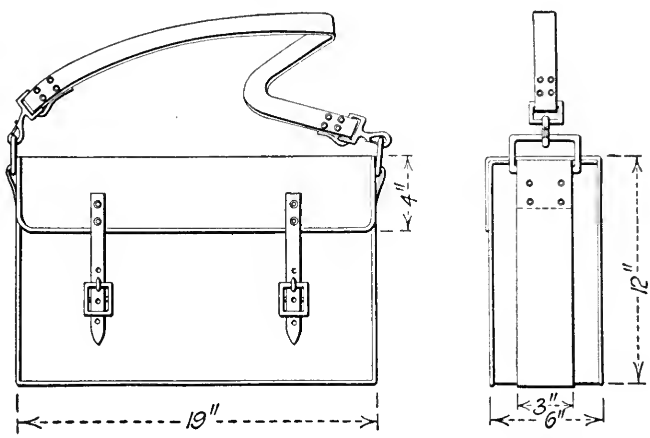


FIG. 40.—CANVAS BAG TO HOLD 50 LB. OF DYNAMITE.

secured by two straps and buckles covers the top of the bag. All seams and edges are leather bound. The capacity of the bag is about one case of dynamite. By its use the hands are left free for the ladders, while in the stope the powder is readily accessible for distribution.

STORAGE AND THAWING

Electric Heating for Primer House (By E. P. Kennedy).—At the Treadwell mines on Douglas Island, southeastern Alaska, it has always been the custom to make up the primers in houses on the surface. There were three such primer houses, all heated by steam. Recently a central primer house was built, in which the primers of all the mines are made. This primer house, illustrated in Fig. 41, is situated 1600 ft. from the nearest boilers and about 1 mile from the Treadwell mine. Four

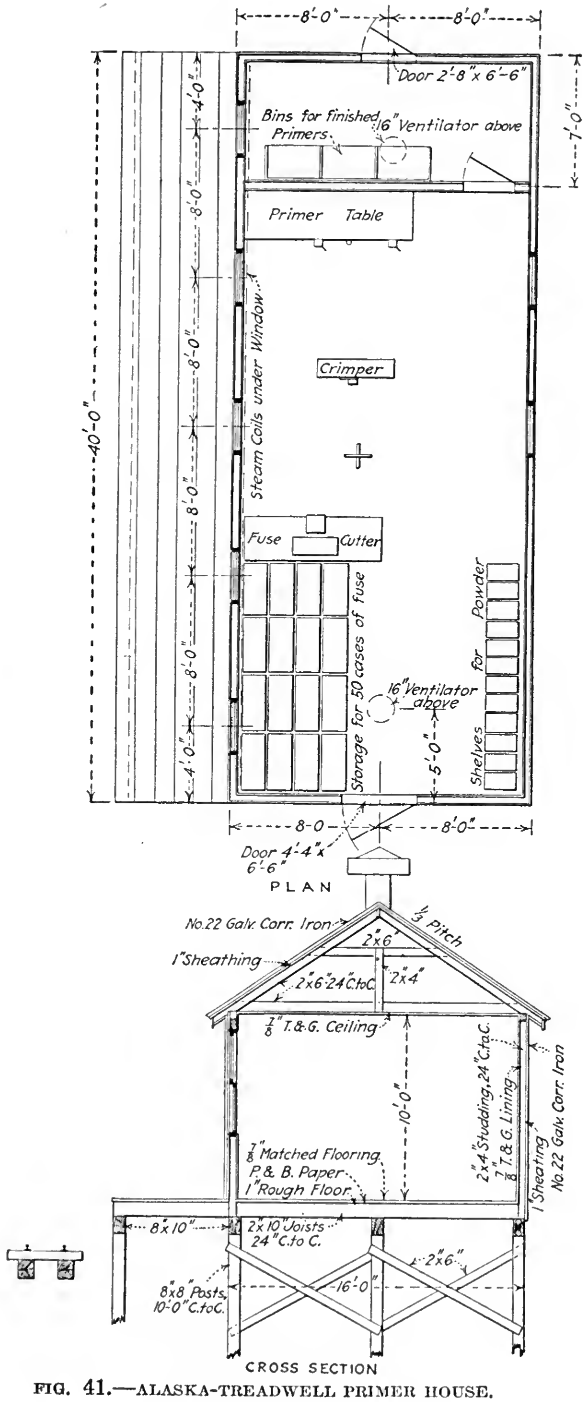


FIG. 41.—ALASKA-TREADWELL PRIMER HOUSE.

sectional steam radiators were installed, but instead of connecting with steam, a 3000-watt Westinghouse bayonet heater was screwed into the bottom of each radiator. A short pipe was tapped into the top of the radiator and connected with a small covered tank, which was kept full of water, a small amount being added when necessary to take care of the evaporation. All leads entering the building and connecting with the bayonet heaters are in conduit and all switches are situated outside the building. The four radiators proved ample for the coldest weather and the amount of radiation can be regulated by opening the switch for one or more of the radiators. The installation has proved most satisfactory. From the point of safety it appears to be above criticism and as there is no loss of energy it would probably prove more economical for an isolated place than a long steam line from which there is continual leakage and radiation.

Each primer consists of a 4-in. stick 1¼ in. in diameter of 70 per cent. nitroglycerin powder; the caps are 6X and the fuse is securely tied to the primer. In the course of one year there are 875 miles of fuse cut up into 2-, 6- and 9-ft. lengths, capped, inserted and securely fastened into 877,000 sticks of powder. One man in charge and two natives can keep all the mines supplied with primers.

Magazine for Storing and Thawing (By Claude T. Rice).—In Fig. 42 are shown the details of construction of the magazine used by the St. Louis Smelting & Refining Co. In choosing a site for this magazine a hollow was selected; hence there is practically no possibility of shooting into the magazine except by malicious intent. The site is several hundred yards away from any other building. The dynamite is thawed in the boxes without any unnecessary handling. This makes the thawing cheaper and less dangerous, for at every point where dynamite is handled there is a source of danger, while the headache that results from unpacking and repacking is not desirable. Thawing houses in which it is necessary to take the dynamite out of the boxes pay only when the amount used each day is small. The thawing house for a large mine should be large enough to hold at least a week's supply, since then the thawing can be done slowly. Consequently, it is just as well to do the thawing in the magazine itself as in a separate place.

It will be noticed from the drawings that there is a partition P of 2-in. planks carried on studding of 2 × 6-in. lumber built out 3 ft. from the walls, inside which the dynamite, still in the original boxes, is piled. To enable air to circulate around them, the boxes are piled with ½ × 2-in. strips between. A small space is also left between the vertical rows of the piles. As the air is free to circulate it requires only about three days to thaw frozen dynamite in the box when the temperature of the thawing room is kept at 70° F., while one winter with the thermometer

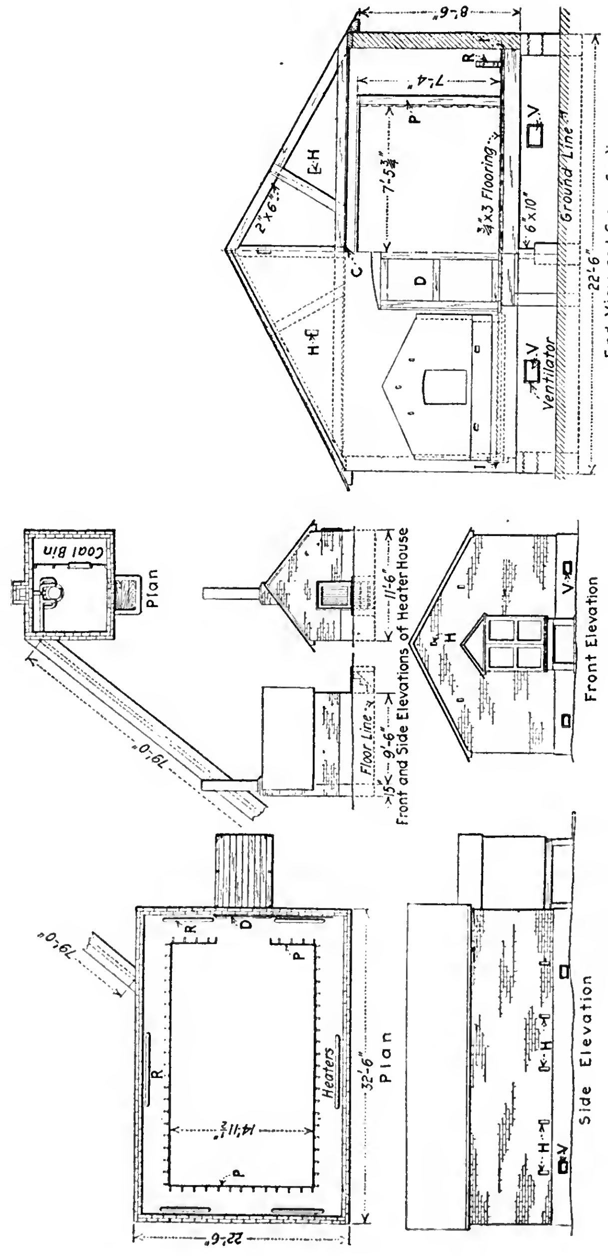


FIG. 42.—POWDER HOUSE OF THE ST. LOUIS SMELTING & REFINING CO.

outside 15° below zero and the magazine around 55°, it took only about five days to do the thawing. With the boxes piled in this manner there is room for storing about 40 tons of dynamite.

Hot water is used for heating. Owing to the slatted partitions and the fact that the radiators *R* are placed close along the sides of the building, the dynamite can be placed within 3 ft. of the water pipe. The hot water is generated in a Tabasco No. 21 heater which is directly connected to the water pipes without any temperature-regulating mechanism on the heater, for it is impossible to heat the magazine above a temperature of 70° in ordinary winter weather. The heater house is placed 79 ft. away from the magazine. The hot water is carried to the magazine in pipes incased in asbestos pipe covering. Anthracite is used to fire the heater.

Both the heater building and the magazine are placed on concrete foundations. The walls of both are made of brick and corrugated iron is used for the roofing laid on top of 1-in. planking. A ceiling *C* of 1 × 3-in. lumber is carried across the building even with the tops of the side walls so that ample air space is left above the dynamite to keep the magazine cool in summer. This air space is connected with air holes *H*, but in winter these are kept closed. The floor is made of 1 × 3-in. planks, and there are air holes *V* connecting with the space under the floor as well as others *I*, coming in just at the level of the floor. These air spaces are arranged so that it would be difficult for a malicious person to insert a burning piece of oily waste in the magazine. Owing to the ceiling, it is impossible to pile the dynamite higher than the brick walls that protect it from bullets. To prevent chilling the magazine while dynamite is being put in or taken out, double doors are provided. These are lined with sheet iron to prevent the entrance of bullets.

According to the Missouri law the dynamite supply in the mine may not be more than enough to last 24 hr. Consequently, the dynamite has to be taken from the magazine to the different shafts each day. This is done by placing the boxes of thawed dynamite in a transfer box, built with double walls, of 2-in. planks so as to provide an air space 2 in. wide all around the box. This box has low wheels attached to it and travels along a track of 1 × 6-in. iron nailed to the floor of the magazine and extending out on the covered porch or entry; this is arranged so that it just comes even with the bed of an ordinary wagon in which the dynamite is hauled to the several shafts. This transfer box is made so as to hold about 30 boxes of dynamite, which is ample supply for one day at these mines. The dynamite does not freeze in the box even with the thermometer down to 16° below zero in the open. At the several shafts the number of boxes of dynamite that are required are taken out of the transfer box and the cover replaced instantly. This thawed dynamite is at

once placed on the cage and lowered to the underground magazines before it has a chance to freeze. The transfer box when empty is returned to the magazine and placed inside it again so that it will be warm when the dynamite is loaded into it next day. This magazine was built according to the directions of George A. Allen, of the Aetna Powder Co.

Around the magazine and at a suitable distance is a picket fence 6 ft. high. Between the fence and the magazine and outside the fence for a distance of 10 ft. is a deep layer of tailings designed to prevent any brush fires from reaching the magazine.

Suitable Powder Magazines.—A dugout in the side of a hill is often considered a good place in which to store explosives. This, however, is a mistake, and such a magazine is ultimately an expensive one, because the efficiency of the explosive, on account of its moisture-absorbing properties, falls off rapidly when stored underground for any length of time. Powder magazines should be constructed, and a good material is brick. Stone or solid concrete or similar materials, which in the event of an explosion would be converted into projectiles, and do damage to surrounding property, should not be used. Brick is pulverized into dust by a dynamite explosion and does no harm. Powder magazines should have bullet-proof doors, and where this is not feasible the doorways should be protected by a screen or barricade. It is advisable to surround the entire magazine with a similar barricade. It requires 11 in. of sand to stop a bullet fired at ordinary range from a modern high-power rifle using smokeless powder, such as is commonly used now by hunters. As has often been suggested before, to sink the entire building behind a heavy earthen parapet is the safest method. Then anything which can be converted into a projectile by an explosion must travel with a high trajectory in order to do any damage.

Electric Powder Thawer, Underground (By A. J. Hewitt).—A suitable place for a powder-thawing house is often difficult to find, particularly after the prospecting stage is passed and the plant begins to extend so as to cover the surface with buildings, some of which are sure to be in too close proximity to the surface thawing house. This was the case at the property of the Beaver Consolidated Mines, Ltd., of Cobalt. After considerable study and careful consideration the following solution of the difficulty was adopted: In the underground workings of the mine, on the 200-ft. level, there was an unused drift, the heading of which was about 800 ft. from the main working shaft. In this heading a powder-thawing cabinet was situated and as heating with steam was out of the question, owing to condensation, a small inexpensive electric heater was installed.

The construction of the thawing cabinet is simple, as can be readily seen; Fig. 43 shows the front elevation, Fig. 44 the side and Fig. 45 the

plan. The cabinet is composed of three tiers of racks, each rack being large enough to contain 60 sticks of dynamite. The racks are made with slat bottoms, the slats being close enough together so that it would be impossible for a stick of powder to fall through, at the same time permitting the warm air to come into contact with the under layer of powder in the rack. There is also an air space left at the sides of each tier of racks

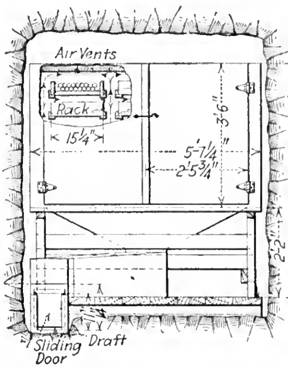


FIG. 43.—FRONT ELEVATION.

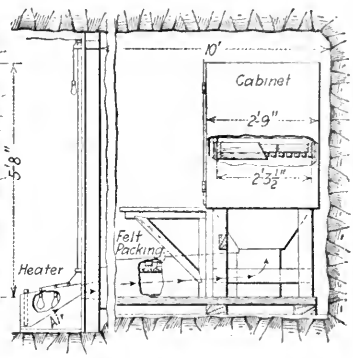


FIG. 44.—SIDE ELEVATION.

to allow the heat to ascend and warm the entire interior of the cabinet. Warm air is conveyed to the thawing cabinet through an ordinary stove pipe well wrapped with 1-in. hair felt and cased with 1-in. lumber. This air duct enters the cabinet bottom at the center, as shown in Fig. 44. Small vents are present in the top of the cabinet to create a draft, drawing warm air continually from the heater. The electric heater is a wooden

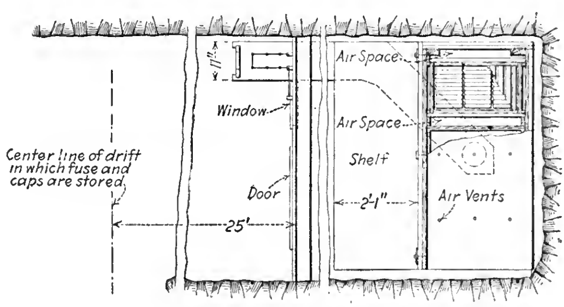


FIG. 45.—PLAN

UNDERGROUND ELECTRIC THAWER, BEAVER CONSOLIDATED MINES.

box covered with zinc, in which are placed six 32-c.p. lamps. One end of the box is made to slide, and is used to regulate the draft in a way similar to the front damper of an old-fashioned box stove. In the other end of the box near the top a hole is made for the stove pipe, which extends about 8 ft. to connect with the thawing cabinet.

The thawing cabinet is situated at the face and about 10 ft. back there is a tight partition. Outside of this partition the electric heater and electric light are installed, all wiring being carefully insulated and none extending through the partition. The electric light is allowed to shine into the thawing room through a small glass window. Access to the thawing room is obtained through a door in the partition. About 25 ft. back from the thawing room in a short crosscut there is partitioned off a small room in which a daily supply of fuse and detonators is kept. In this

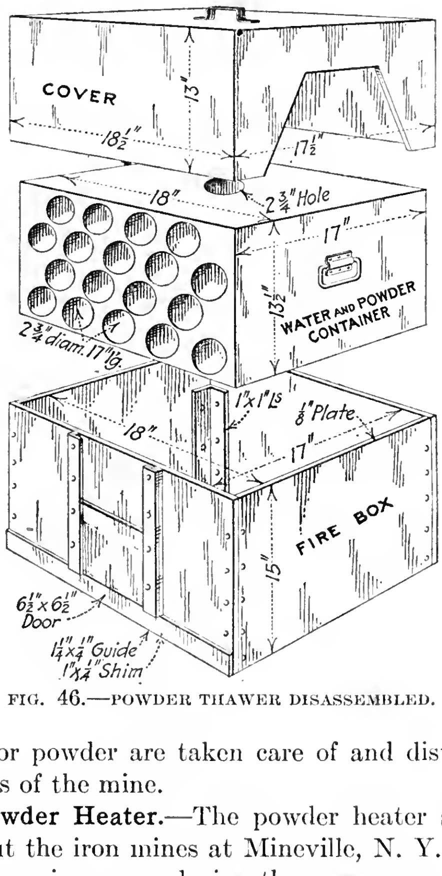


FIG. 46.—POWDER THAWER DISASSEMBLED.

room all orders for powder are taken care of and distribution made to the different parts of the mine.

Hot-water Powder Heater.—The powder heater shown in Fig. 46 is used throughout the iron mines at Mineville, N. Y. Due to the low temperature in the mines even during the summer months, the powder must be thawed except for a certain percentage of nonfreezing powder. In using the heater shown, the fire is first kindled in the firebox, the water container is placed over the firebox, resting on corner angles, and then the cover is slipped over the water container. The latter is 17 × 18 in. by 15 in. high and made on the principle of a return-tubular boiler. Flues, 18 in number and $2\frac{3}{4}$ in. in diameter, are fitted in two opposite

faces of the container; each will hold six sticks of $1\frac{1}{8} \times$ 8-in. powder, making the capacity of the box 108 sticks. Water is poured in the top of the container until it is well above the top row of tubes. After the water has been heated to a temperature which the hand will bear, the container and cover are removed to a safe distance from the firebox and the fire completely extinguished. The cover is then removed, the flues are filled with the sticks of powder and the cover replaced. In a short time the powder will be sufficiently soft for use.

Manure Powder Thawer.—A satisfactory arrangement, utilizing the heating power of fresh horse manure for thawing frozen dynamite, is used underground at Mineville, N. Y., by Witherbee, Sherman & Co., Inc. The thawer, as shown in Fig. 47, has a capacity of eighteen 50-lb. cases, and is nothing more than double box built of 2-in. planks with an

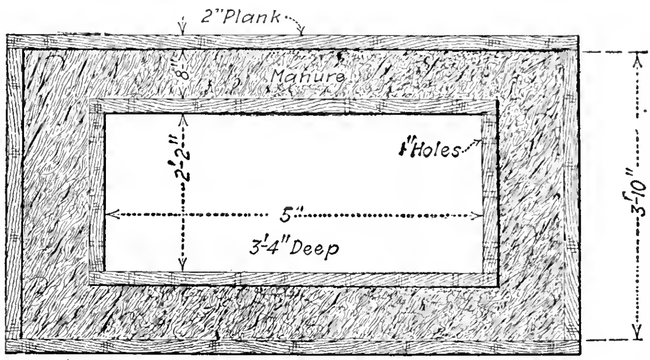


FIG. 47.—HORIZONTAL SECTION THROUGH MANURE-WARMED THAWING BOX.

8-in. space between the outer and inner boxes, this space being filled with manure. A double lid, also filled with manure, is used and the bottom of the inner box is similarly filled, so that the powder is surrounded on all sides, top and bottom by the heating material. One-inch holes are bored through the inner box into the manure-filled spaces to permit circulation of the heated air. The powder may be put into the thawer while in its original boxes or the sticks may be put in loosely, which enables it to thaw a little more quickly. Where loose sticks are placed in the thawer and different strengths of powder are used, it will be found convenient to make vertical partitions in the inner box, so that the different grades will not become mixed. It has been found that when the manure is fresh, the temperature of the inner box may become as high as 110° F. This arrangement does away with all the dangers and inconveniences of any other type of thawer and has been found satisfactory.

SAFETY PRECAUTIONS

Methods of Making Primers (By William W. Jones).—No entirely satisfactory method of constructing a dynamite primer has heretofore been available. Of the various methods of inserting the cap in the cartridge, some of the most unsatisfactory are here illustrated, Fig. 48. In 1 and 2, the fuse is shown laced through the cartridge. The lacing of 1 is perhaps the more objectionable of the two. The following disadvantages, however, attend either method: (1) The dynamite is likely to ignite as a result of the powder train in the fuse burning through its covering at some point in the cartridge and igniting the latter, which causes

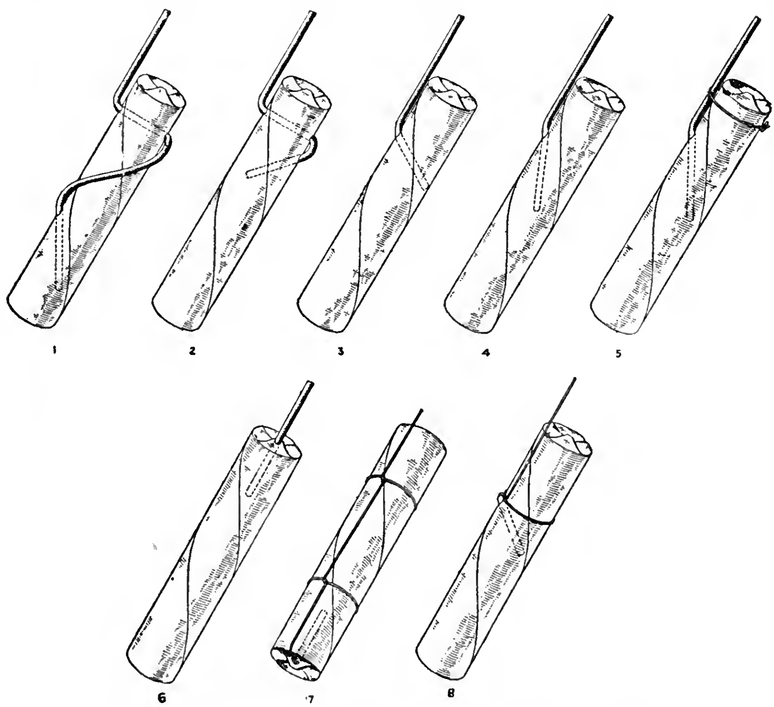


FIG. 48.—UNSATISFACTORY METHODS OF MAKING PRIMERS.

an imperfect explosion; (2) the powder train is likely to break where the fuse is bent at an acute angle, causing a misfire; (3) the diameter of the primer is so increased that it cannot always be pressed down on the rest of the charge and the resulting air gap may intercept the transmission of the explosion so that the inner part of the charge is not exploded; (4) the cap does not point along the charge and so loses part of its efficiency, and if thrust in too far, it is liable to penetrate the opposite side of the cartridge and be exploded by scraping on the rock.

In 3 and 4, the cap inserted at an angle does not give the most efficient detonation, and not being tied, it can easily pull out. In 3, either end may project, and thus cause a premature explosion. The same objection applies to 5, except that the tying tends to prevent the cap from pulling out. Tying is often neglected, however, as it takes time, and string is not always handy. In 6, the detonator points in the proper direction, but not being tied, is easily pulled out. In this case, the ends of the paper wrapping can be unfolded and tied around the fuse with a string, but as stated, tying is frequently neglected.

In 7 is shown perhaps the most commonly used way of priming with an electric detonator; it is open, however, to two objections: The business end of the detonator, if the primer be on top as usual, points outward instead of into the bulk of the charge, and the half-hitches are likely to damage the insulation. In 8, the detonator does not point to the

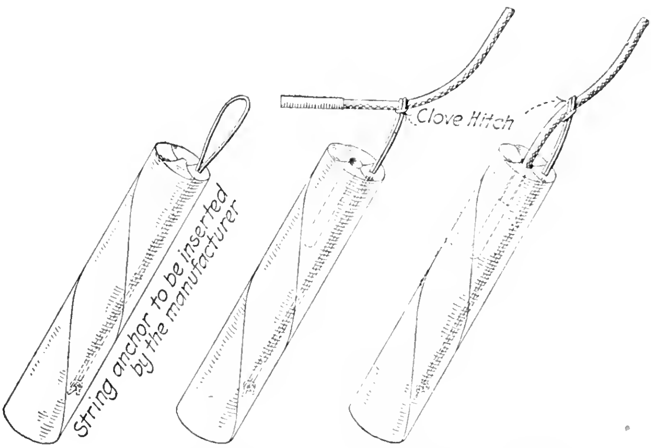


FIG. 49.—NEW METHOD OF CONSTRUCTING PRIMER.

best advantage and is liable to project on one side. Furthermore, in tightening the half-hitch around the cartridge, the sulphur plug may pull out of the detonator and cause an explosion.

It will be noted that the principal objections to the methods of priming here illustrated are that the cap is not placed to do its work most efficiently, or it is likely to be pulled out, causing a misfire, or it is liable to premature explosion. A recently invented and patented device illustrated in Fig. 49 will largely do away with these objections and should be a great aid in reducing the number of explosive accidents. It permits the insertion of the cap in the most efficient manner, while practically eliminating the danger of pulling it out or of breaking the fuse or wires. It consists of an anchoring device, a piece of string with a knot on the end, to take the pull on the fuse or wires and relieve the cap itself of

tension. A double cord is used and a clove hitch taken over the fuse or the wires. The method of doing this is evident from the drawing. The cord must be included in the powder stick when it is manufactured. It is evident that in using this device there is little or no danger of the caps pulling loose from the cartridge, and this simple fact makes it possible to place the cap in the safest and most efficient position.

Blasting Bulletin Board.—The accompanying form, Fig. 50, shows the blasting bulletin board used at the shaft stations of the St. Louis Smelting & Refining Co. in the Flat River district of Missouri. As the men go off shift, they mark in the proper place on the record on the bulletin the holes just fired. The driller is designated by name as well as by the number of his machine. Thus a driller on the next shift knows by the number of the machine, if he does not know the name of the man using his machine on the other shift, the number and kind of holes that were blasted in his working place on the preceding shift.

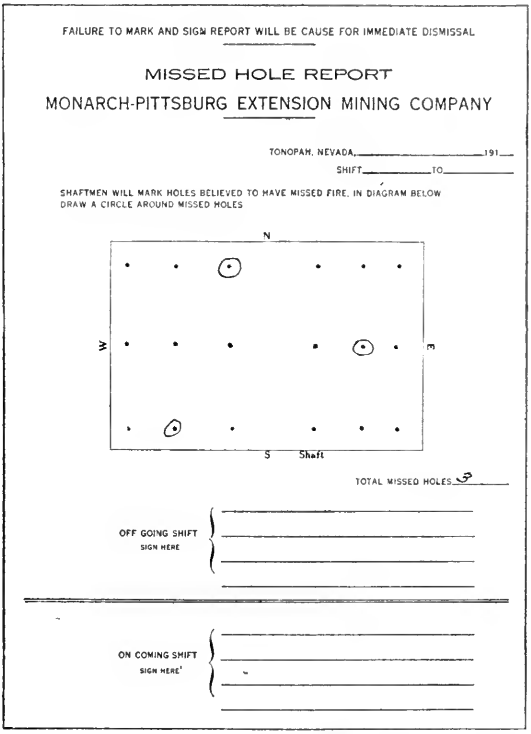
BLASTING BULLETIN BOARD								
Shift Boss.......................... Holes Fired					Shift Boss.................... Holes Fired			
Breast	Stope	Lifter	Name of Driller	Number of Drill	Name of Driller	Breast	Stope	Lifter
Special Instructions								

It is the duty of the drillers and backhands to study this bulletin carefully before going to work and upon arriving at their respective places of work to make a thorough search for missed holes.
All missed holes must be fired before drilling is started.

FIG. 50.—ST. LOUIS SMELTING & REFINING CO. BLASTING REPORT.

The method of designating the holes will differ according to the method of breaking ore. The object in giving the names of the holes is that by such a system the men can take care against drilling into missed holes. When so many men are blasting in the mine it is impossible to count the holes and determine those of each man. Even when it is impossible for each miner to stop in a place where he can tell from the reports whether his are the holes exploding, it is well to make each driller chalk under such a bulletin board the kind of holes that he has fired and the number. It makes the work of the driller safer, and less time is lost by the miner on the next shift in looking over the breast to make sure that there are no cut-offs or missed holes.

Form for Missed-hole Reports (By B. H. Smith).—An ingenious method for reducing the probability of missed-hole accidents was adopted in sinking the Monarch-Pittsburgh shaft, at Tonopah. The idea is not entirely new, having been used in the Cœur d'Alenes previously. A blank form, as shown in Fig. 51, was placed in a conspicuous place of the change house, and in the rectangle a plan of the holes in the shaft bottom was made. The shift coming off after blasting, marked rings around the dots representing the holes which it was suspected had missed, and each



FAILURE TO MARK AND SIGN REPORT WILL BE CAUSE FOR IMMEDIATE DISMISSAL

MISSED HOLE REPORT

MONARCH-PITTSBURG EXTENSION MINING COMPANY

TONOPAH, NEVADA, ____________ 191__

SHIFT ________ TO ________

SHAFTMEN WILL MARK HOLES BELIEVED TO HAVE MISSED FIRE, IN DIAGRAM BELOW
DRAW A CIRCLE AROUND MISSED HOLES

TOTAL MISSED HOLES 3

OFF GOING SHIFT
SIGN HERE

ON COMING SHIFT
SIGN HERE

FIG. 51.—FORM FILLED OUT TO SHOW THREE HOLES MISSED.

member of the shift signed below. The shift coming on also signed at the bottom as an indication that each member had examined the diagram. Two purposes were served by this device. (1) The new shift was forced to receive information of the possibly dangerous holes; (2) the liability of the company to damage suits was decreased, since the signatures of the men would show that they were aware of the presence of the danger. The usual method of leaving the record of suspected holes on a blackboard for the new shift to read, is never satisfactory. The shift which has just blasted is usually in too much of a hurry to chalk up the missed

holes and the shift coming on frequently does not bother to read anything that has been written.

Neutralizing Blasting Fumes (By W. H. Mawdsley).—The sulphur fumes, caused by the ignition of pyritic ore in blasting, are in some mines exceptionally severe; in such cases the charges are usually fired only at intervals when work in other parts of the mine has ceased. A simple and effective way of overcoming this difficulty, as well as that caused by the fumes from the explosive itself, is by the use of suitable chemicals as tamping. The alkaline hydrates give the desired results and also absorb some of the CO_2; the addition of an oxidizing agent in the tamping converts the CO into CO_2 which can then be absorbed. With regard to sulphur fumes, moist slaked lime has proved invariably effective. A small quantity is placed at the bottom of the hole and more is used on top of the charge in place of the usual clay "cocks." At the instant of explosion the hydrate comes in contact with the nascent gases and immediately absorbs them. Another method is first to pour a little water into the hole and then add some soluble hydrate (such as barium or strontium) which is dissolved in the water; the charge is now introduced, the solution being thus mixed in with the explosive. A further quantity of hydrate is used above the charge in place of the ordinary tamping. The material is made into suitably sized cartridges covered with as little paper or organic matter as possible. This tamping is a convenience to the miner, and by preventing the sulphur fumes, makes blasting in pyritic ore possible at all times. By the addition of an oxidizing agent, the carbon monoxide given off by some explosives is to a large extent rendered innocuous.

III

ROCK DRILLS

Drilling Kinks—Machine Supports—Maintenance

DRILLING KINKS

Drilling Mesabi Gopher Holes.—Blasting in the Mesabi openpits is necessary in stripping if the overburden is somewhat consolidated, and rather general in mining unless the ore is unusually soft. Two methods of drilling and blasting are in use. One consists of drilling deep downholes at the top of the bench, using a jumper drill, often with several men, chambering the bottom with dynamite and blasting with black powder; the other is given the wholly indefinite name of "gopher-holing," a term which has a different signification in every mining region.

Gopher-holing here consists of working out an inclined hole about 16 to 20 ft. deep, beginning at the bottom corner of the bank and extending in at an angle of 10 to 20° below the horizontal. This is accomplished in various manners. In the cases observed, a set of tools similar to that illustrated in Fig. 52 was employed. In addition to these, an ordinary hand auger was used to start the hole and take out the first few feet; this hole was blasted with a stick or two of dynamite, the effect being to leave a long bootleg of 8 to 14 in. in diameter. The auger being unwieldy in deep holes, a long moil-pointed bar usually of 1¼-in. steel was next brought into service, shown in the illustration. This was driven into the bottom of the bootleg for a foot or two, by two men with doublejacks. When progress became slow, a perforated plate about 1½ × 6 × 8 in. was slipped over the end of the moil and wedged to it, either with two small wedges or with one wedge and a track spike as illustrated. By hitting the head of the wedge with the two doublejacks, the moil was extracted. During this process, it was usually supported on a log as shown, to keep the plate off the ground.

The moil hole was then blasted by inserting a stick or two of powder and exploding with an electric cap. These alternating processes were continued until the desired depth of hole was obtained, the average diameter being perhaps 12 in. This intermediate blasting was done with a single dry cell. It was somewhat surprising at first to see the miner load his hole, step a foot or two to one side of the collar, connect his battery,

and set it off. As a matter of fact, material was not even shot out of the mouth of the hole, the sole effect being to shatter the ground adjacent to the charge. To remove the broken material, the long-handled spoon shown was used. It is made of an ordinary No. 2 shovel by bending the sides up straight so as to give a depth of about 2 in. and opening the socket to take the large 2- or 3-in. end of a peeled sapling. When the hole was sufficiently deep, a larger charge of dynamite, 10 to 13 sticks, was exploded in a bottom to give a good chamber. Black powder to the extent of about three kegs was charged into the chamber by shaking

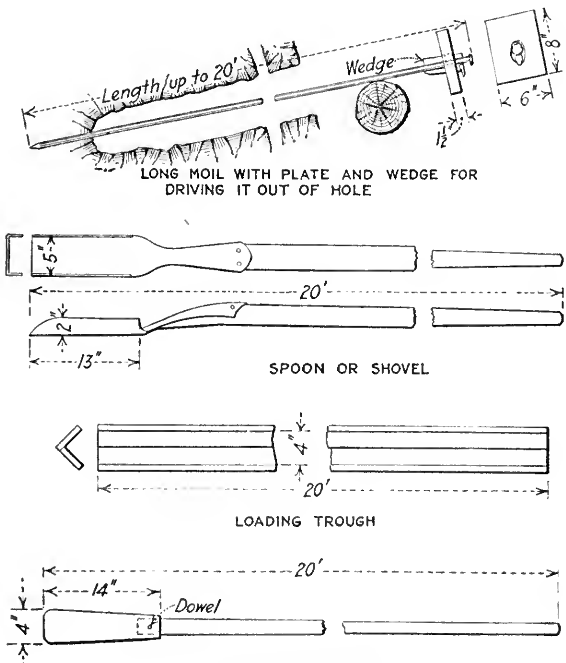


FIG. 52.—TYPICAL SET OF GOPHER-HOLE TOOLS.

it down the launder illustrated. A dynamite primer was inserted in the middle of the powder charge and the whole tamped tight with the tamping bar shown in the illustration. The remainder of the hole was then also tamped with lean ore. The tamping bar consists of a round, tapered wooden head about 4 in. in diameter at its big end and 14 in. long, with the smaller end bored to permit the insertion of a 2-in. sapling which is fastened with a wooden dowel pin. No nails are used, and in the construction of the trough, copper nails are considered advisable. It is an obvious rule of safety not to bring iron and rock into contact in the presence of black powder.

The holes were blasted in groups and for this purpose the battery was discarded and a push machine used. It is customary to move some distance off for this blasting, but if it be well carried out, one might stand on top of the blast without injury, since there is rarely any discharge of material, the whole bank being lifted and allowed to drop, thus effectually shattering it.

Cutting Mass Copper.—In the Lake Superior district a great deal of mechanical cutting of copper is done underground in order to reduce the larger masses of native metal to sizes which will readily go into the skips. The process does not differ essentially from that of cutting solid copper in any form, and as frozen masses of this metal are of frequent unwelcome occurrence in metallurgical works, a description of the method will perhaps be of interest. As used at present, the method is essentially the same as that employed 70 years and more ago, the difference being that pneumatic hammers rather than hand sledges are used to strike the cutting

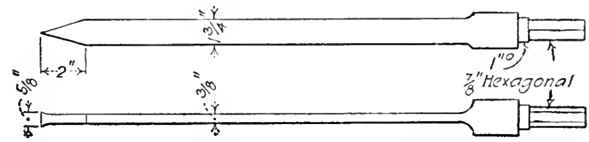


FIG. 53.—COPPER-CUTTING TOOL FOR PNEUMATIC HAMMER.

tool. The general dimensions and the shape of this cutting tool are shown in Fig. 53; its length when new may be as much as 4 ft., and it is used until its length has been reduced by continued sharpening to a few inches. It is used in a pneumatic hammer of the size and style usually employed in riveting operations.

The operation of cutting a large mass is as follows: After the mass has been freed from its resting place in the rock, and is lying loose on the foot wall, it is marked off by the shaft captain along the lines which will most advantageously divide it into smaller pieces. Grooves are then cut along these lines by taking out successive triangular chips; when cutting is being done by a skilled operator, each chip is continuous through the entire thickness of the mass. It is much easier to have the chips thin out to an edge on one side than it is to take out a flat strip, the tool being reversed at each cut. The width of the tool corresponds closely to that of the cutting edge of the chisel. The hammering action serves to compress the copper in the chip, so that the latter is about one-half the length of the groove from which it came and is correspondingly large in section. The chisels will cut copper for a considerable time without dulling, but they dull rapidly on the pieces of included rock which are frequently encountered. The process is slow and the work tells on the wrist and arms of the operator, as the tool must be held closely to the

copper. It is usual for men to work in pairs, one resting while the other takes a cut through the mass. Two men working in this way under ordinary conditions will probably cut 1 sq. ft. of area per shift, making the labor cost approximately $6 per square foot, which is about one-half of the cost with the old method when one man held the chisel for two men striking alternately.

The opinion has popularly existed in the minds of many that copper may readily be cut by the oxyacetylene process. Various inquiries have led to the conclusion that this process is entirely inapplicable, at least in a way similar to that in which it is applied in cutting steel. It will be remembered that the latter is essentially an oxidizing process, steel oxidizing very rapidly in an atmosphere of oxygen when heated to the temperature of the oxyacetylene flame. Copper does not oxidize readily in this way, and its high heat conductivity is also a factor as preventing the localization of the heat. It is not known that the method has been tried out locally on mass copper, but it has been tried at refineries by men expert in its use and these attempts, so far as known, have resulted in flat failures.

Machine-driven Auger for Soft Ground.—On the Lake Superior iron ranges it has been customary to drill the holes in certain soft ores with

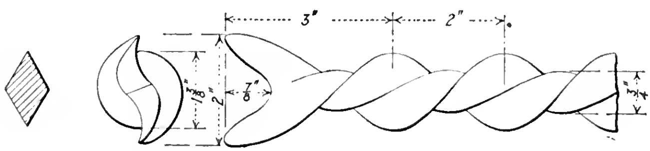


FIG. 54.—BAR SECTION AND FORMED BIT.

hand augers. It is now becoming rather general practice to use an auger bit in a hand-held rotating hammer machine. Soft ground has been usually considered unsuitable for this type of machine, inasmuch as the steel tended to bury itself. It is evident that with an auger bit the difficulty is avoided, since the action of the twisted steel is that of a screw conveyor, rapidly clearing the hole of all cuttings. The rotation of the machine has a certain amount of drilling effect, but the speed of drilling is greatly increased by the hammering of the piston on the steel. A further advantage lies in the fact that when a hard rib is encountered, the regulation rose-bitted hollow steel can be substituted for the auger until the soft material comes in again.

The success of the machines is such that they are now made specially designed for the work, having a higher speed of rotation and a lighter blow. The steel of which the bits are made is not twisted in the shop as in the case of hand augers, but comes twisted from the manufacturer. The original bar has a diamond cross-section similar to that shown in

Fig. 54. It is twisted so as to give one complete turn per 4 in. The character of the bit forged by the blacksmith is also shown approximately. The greatest diameter of the twisted bar is about 1⅜ in. The bit of the starter has an overall diameter of 2 in. The other pieces of the set decrease about ¼ in. in gage and increase about 18 in. in length. The shank end is welded on in the shop; it is hexagonal and has a small collar to facilitate removal of the steel from the hole.

Removing Broken Drill from Hole (By George A. Addy).—An ingenious and useful scheme for removing the bit end of a broken drill steel from a hole is illustrated in Fig. 55. It consists simply of a ring of square iron *A* of a suitable diameter to slip over the broken steel. Two holes *B* are drilled on opposite sides of this ring and a stout cord passed

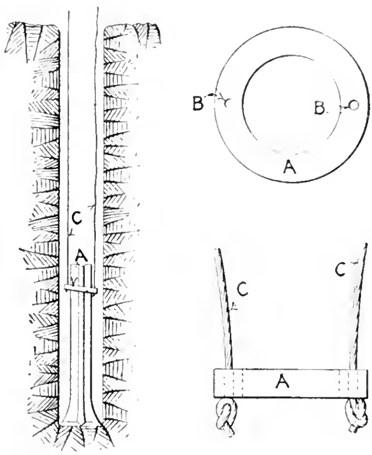


FIG. 55.—RING AND CORDS FOR EXTRACTING BROKEN BITS.

through these holes and knotted on the lower sides. The ring is slipped over the steel in a down-hole, and by loosening one cord and pulling on the other, the square corners of the ring grip the steel so that it may be removed. In case it is desired to use the device in a flat-hole, one of the holes *B* is drilled large enough to take the pointed end of a scraper, and this tool is used in slipping the ring over the steel.

MACHINE SUPPORTS

Pneumatic Drill Column (By Sven V. Bergh).—An important factor to be considered in getting the highest efficiency out of the modern hammer drill is the simplification of the drill mounting when such mounting is required. The labor cost, compared with the other costs of operation, is in most cases predominant, and it must be borne in mind that,

roughly estimated, only 40 to 50 per cent. of the total shift can be used for drilling, the rest being absorbed in setting up and other manipulation of the machine. When a column or bar has to be used, the drill may be either attached to it directly or it may be carried on an arm; the column itself may be of either the single-screw or the double-screw type. In any case, its height is adjustable to only a limited extent. Hence, to make a set-up in the best position for the round of holes to be drilled, the workman has to use a good deal of skill and do considerable planning; in most cases a lot of time is wasted in mucking, arranging blocking, picking down back or picking up bottom, in order to get good bearing surfaces for the column.

To facilitate these operations, a pneumatic drill column was devised and has been used with good results for several years at the iron mines of Malmberget, Sweden. It resembles in general the telescopic air-feed of the stoper type of drill. A longitudinal section is shown in Fig. 56. The height of this particular column can be varied between 6.23 and

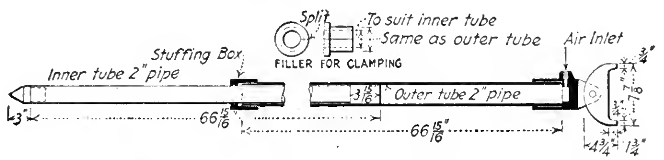


FIG. 56.—COMPRESSED-AIR TELESCOPIC MOUNTING FOR LIGHT DRILL.

11.48 ft. When the column is fully run out, the compressed air is shut off by means of an automatic cut-off valve. There is no need of the usual arm in order to provide a sufficiently wide range of positions for the machine. The column, drill and all are easily removed from one position to another as required, and the absence of an arm makes this all the easier. The drill clamp is adjustable with one single screw-bolt, and is made with a hinge so that the machine can be swung over to one side when the drill steel has to be changed. The clamp is attached to the outer tube. This has a toothed foot-piece which bears on the bottom and resists rotation. The inner tube ends in a point which bears against the back. It is, of course, not subject to any torsion.

The column was found sufficiently stable when the regular drilling pressure was used, but is adapted only to light hammer drills of the stoper type with telescopic air-feed and for drilling in soft to medium-hard rock. It should be noted that when air is turned off, the machine first stops drilling and the air in the column maintains static pressure for some time. It can, however, be exhausted rapidly if desired.

Tripod for Handling Long Drill Steel (By Le B. Reifsneider).—A device has been found useful by the Spanish-American Iron Co. for handling long pieces of steel when drilling vertical holes up to 35 ft. in depth,

using tripod drills. It consists of the parts of an old tripod so badly worn as to be no longer safe as a drill mounting. The usual short pipe legs are removed and replaced with lengths of $1\frac{1}{2}$-in. pipe, the latter being threaded and having a coupling on the lower end for convenience in adding short lengths of pipe when it is desired to increase the height of the rig. An eye-bolt is put into the opening in the saddle so as to hang down and is secured there by a washer and nut, and a single block with a $\frac{3}{4}$-in. manila line rove through it is hooked into the eye-bolt. The line is two and a half times the length of the tripod legs, and at one end a simple straight hook of $\frac{1}{2}$-in. round iron is fastened.

In use this modified tripod is first laid out on the ground, the front legs are spread to a width that will make a stable base, usually about one-half the height although sometimes less, and all bolts are drawn up and tightened except the nut on the through bolt which holds the rear leg. The block is then hooked into the eye-bolt, the rope rove and the tripod raised and moved over the hole where the machine is drilling, so that the hook on the rope falls approximately over the center of the hole. In removing a piece of steel from the hole it is first pulled up as far as it can be conveniently handled with the dolly bars; one man then lets go the dolly bar and takes three turns of the hook end of the rope around the steel close to the collar of the hole, fastening the hook back over the rope above the turns on the steel. He then pulls the rope taut and takes several turns around the rear leg of the small tripod on which the drill is mounted. As soon as he has done this, the other man lets go the dolly bar and aids the man at the rope to pull the steel up until the bit swings clear of the collar. After this the steel is lowered by paying out the turns of the rope on the leg of the drill tripod. The steel is always handled between the two front legs of the rig, since this gives greater stability.

The regular drill crew of two men can raise a piece of steel 35 ft. long, using a tripod 22 ft. high. The rig is used for all steel over 20 ft. in length. Beside the saving of time and labor in handling the steel this is also a valuable safety device, since by using the drill-tripod leg as a snubbing post the steel is always under control and can be handled safely on a narrow ledge in a high wind.

Simple Machine Bar.—In Fig. 57 is shown the design of machine bar used in the mines of the lead district of southeastern Missouri. It is a one-screw post without any header casting at either end. The teeth for gripping the top blocking are made by cutting the pipe used for the column in two by drilling it full of holes. The pipe is then plugged with a piece of oak to keep grit from getting down into the thread of the jacking screw. The illustration shows this block only partly driven in. A collar of $2\frac{1}{2} \times \frac{1}{2}$-in. iron is shrunk on the upper

end to reinforce it after the plug has been hammered in. A similar collar is also shrunk on the bottom after the jack screw has been put in. There is no shoe on the bottom end of the jacking screw. It simply ends in a blunt point. Formerly this was stuck in the axle hole of a car wheel in making a set-up, but the ends often penetrated too far into

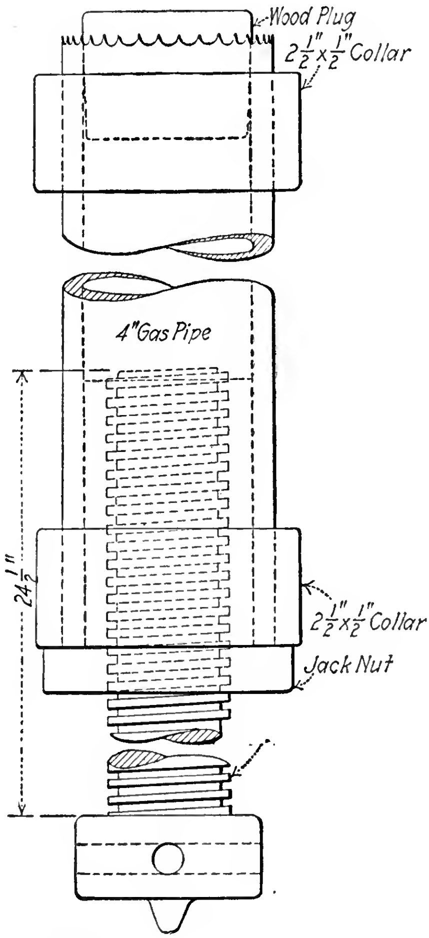


FIG. 57.—MACHINE BAR USED IN SOUTHEASTERN MISSOURI.

the wheel and as the side strain was strong on the jacking screw the result was that several screws would be broken in the course of a month. On that account simple plates about 6 or 8 in. wide and 1 in. thick, with a depression in the center for taking the end of the jack screw, are now provided. These do not let the jack screw enter far enough to put any side strain on the screw. With this shoe plate it is possible to

set up without any trouble on a steeply slanting bottom. If there were a shoe to the jack screw, a considerable side strain would be thrown on the screw unless a block was put under it to level up with such a sloping bottom in order that the column could be stood perpendicular to the foot blocking. In other words, more care has to be taken in making a set-up when a foot shoe is used than when one is not, and since, therefore, the work of setting up with this type of shoeless column is easier and quicker, the machinemen like it better than the bar of the regulation type. As the column is always set up on solid rock in this district, the fact that the end of the jack screw is not of large area is unimportant.

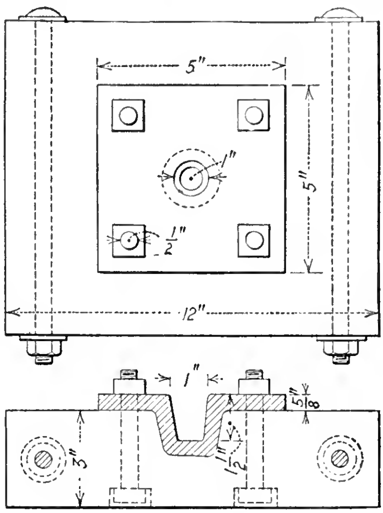


FIG. 58.—SHOE FOR TRIPOD LEG ON BROKEN GROUND.

Steady Tripod Set-up on Loose Rock.—In certain classes of mining, especially with some types of underhand stoping, and where the ore-body is too wide for a column, a tripod set-up for the drills is frequently necessary. Often it is not possible to place the tripod legs on solid rock and the yielding set-up which results interferes with good drilling. When such a set-up is necessary, several expedients are employed to steady the drill and distribute its weight over a large area. In some Missouri mines the tripod is set on old car wheels, but these are heavy to handle and if mixed with the ore might cause serious delays around the gyratory crusher. In Butte wooden blocks are employed, but unless they are at least 5 in. thick or shod with iron, they are easily split by the sharp ends of the tripod leg.

A device used by the Cleveland-Cliffs Iron Mining Co., in Michigan, consists of a 10 × 12-in. piece of 3-in. oak, Fig. 58, shod with a 5-

or 6-in. square of ⅜-in. scrap iron. The shoe, which is fastened to the block by four ½-in. carriage bolts, has a depression about 1 in. in diameter and 1½ in. deep, to receive the tripod leg. To prevent splitting, the block is bound together at each end by a ½-in. bolt.

Steadying Leg for Rock-drill Bars.—In sinking the No. 2 Hancock shaft in the Michigan copper country, the rock drills were mounted on arms carried by a regular shaft bar with a single screw at the end. On each arm a collar was used carrying a leg to steady the bar under the hammer of the three machines which were used at one time, and which otherwise would have made it difficult to keep the bars tight. The details of this steadying leg, with the collar by which it is attached to the

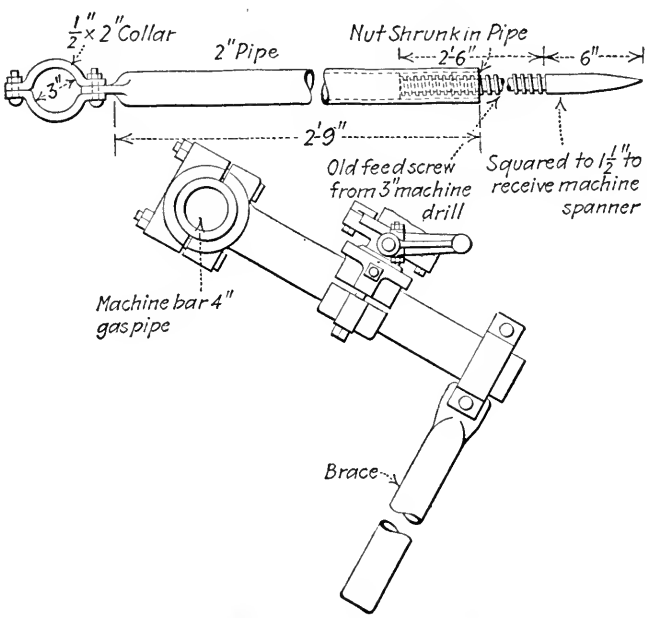


FIG. 59.—DRILL-ARM STEADIER USED IN HANCOCK SHAFT.

arm, are shown in Fig. 59. The device consists simply of a collar with a pipe fastened to it, from the lower end of which extends a screw leg made from an old machine feed-screw. The end of this screw leg is drawn down to a sharp point, which is firmly braced and screwed out, from time to time, so that the machine is kept braced tight from the ground and cannot swing. It is usually the surging of the machine that loosens the bars. This device steadies the arms upon which the machines are directly mounted, and therefore the large shaft bar receives far less jar and vibration than would otherwise be the case.

Wedged Arm for Drill Columns (By R. A. Rule).—The drill-column arm generally used in underground drilling is clamped to the

column by two bolts. In the arm shown in Fig. 60 bolts are not used. The arm is attached to the drill column by a clamp in which a wedge and slotted key are used instead of the usual bolts. The wedge fits into a slot in the key, and by striking the wedge a smart blow with a

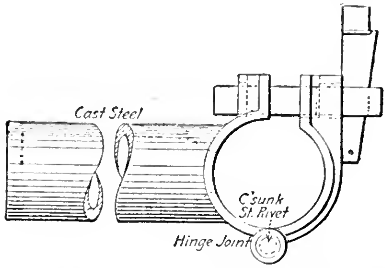


FIG. 60.—WEDGE AND KEY DRILL-COLUMN ARM.

hammer the clamp is drawn tightly about the column. To loosen the clamp the lower end of the wedge is struck with a hammer. It requires but one hand to loosen the wedge; hence the other hand may be used to hold the arm when the grip is released.

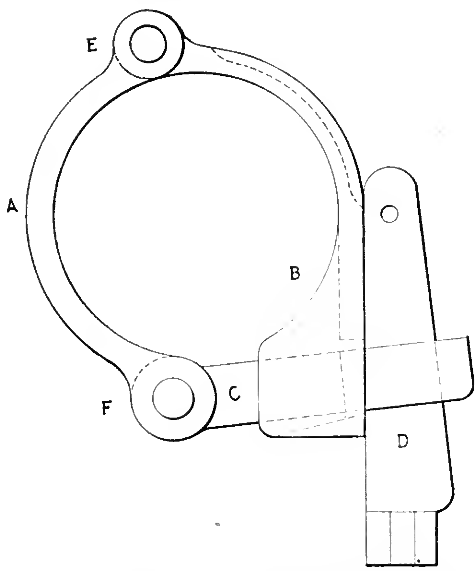


FIG. 61.—HINGED COLLAR FASTENED BY WEDGE.

Wedged Drill-column Collar (By R. A. Rule).—Fig. 61 illustrates a four-piece post collar, which dispenses with the use of bolts. The pieces *B* and *C* are hinged to the piece *A* at *E* and *F*, respectively. The piece *C* passes through a hole in the end of the piece *B* and is itself

slotted to receive the tapered key *D*, which working on the piece *B* forces it against the post. The blow of a hammer on the key *D* thus tightens or releases the clamp. The ease and rapidity of operation of a hinged-and-keyed collar, as compared with the usual bolted collar, results in an appreciable saving of time.

Copper Range Drill Column.—An unusual form of drill column is used in the mines of the Copper Range company in Michigan. This post has two toothed ends. It is mounted on top of a two-screw jack which is fitted with a wooden center block into which the teeth of the bottom of the column nip when the post is set up, Fig. 62. The column and the

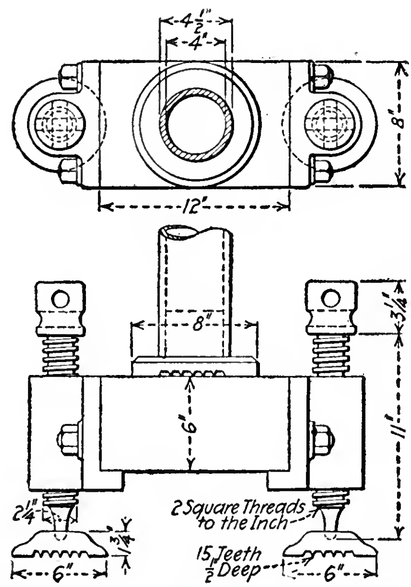


FIG. 62.—TWO-SCREW DRILL COLUMN USED IN COPPER RANGE MINES.

jack are independent and thus the weight is divided. Moreover, one jack can be made to serve several different lengths of column. This is a decided advantage whenever, as at the Copper Range mines, it is necessary to have several lengths of column owing to the different heights of the back above the filling. The advantage of a two-screw jack is obtained without the drawback of the great increase in the weight when it is fastened tightly to the post part. The jack nuts are castings that have a projecting ledge for the wooden block to rest upon. Two bolts go through the block and hold the nut castings and jack block together. This jack block lasts about a year before it becomes so worn that it has to be replaced. The column itself is simply a standard 4-in. gas pipe with toothed head-pieces shrunk tightly into it at both ends. Because of its

cheapness, the men in the stopes are given three lengths. Consequently, they do not have to build up much blocking when making a set-up.

MAINTENANCE

Handling Drill Steel at the Quincy Mine (By L. Hall Goodwin).—The method of handling drill steel at the Quincy mine, Hancock, Mich., is different from that usually employed at the larger mines of the Lake Superior district, where the problem is to sharpen the steel for several widely separated shafts at a centrally located shop. At the Copper Range mines and the Calumet & Hecla the drills are kept loose and are carried to and from the various shafts in steel boxes, each shaft having its special box.

The main distinguishing feature of Quincy practice is that the pieces of steel are kept in slings, each sling containing 8 to 10 pieces, which are assigned to a certain contract. Dull drills are put up in slings at the working place underground, and sharp ones before they leave the shop. The sling consists of a piece of ½-in. flat iron, 1½ in. wide, formed into a ring having an inside diameter of 4 in. The steel is held in this ring by means of three wooden wedges, 8 in. long, placed side by side, the middle one being reversed end for end with respect to the other two, and hammered into place; this forms a firm bundle of convenient size for handling.

All miners have a contract number, although only those employed in mine development actually work on contract; for other miners, the number is used only in distributing tools and supplies. Each piece of drill steel is assigned to a certain contract, and the number of that contract is cut on its shank.

Dull steel is sent to surface and sharp steel sent underground after each shift has gone off. The dull steel is carried to the shaft by the miners and dropped where it can be loaded easily into the south skip, as it has become an arbitrary rule at this mine always to use the south skip for handling tools and supplies. When all men are up, the man-cars are immediately replaced by skips, and the first skip hoisted in the south road picks up the slings of dull steel from all the levels; the sharp drills are sent down on the next trip of the same skip.

At surface, the skip automatically discharges its load of steel at a special dump just above the collar of the shaft. It passes into a concrete chute, the bottom of which is faced with old rails, flange up. This chute starts off at 30°, but its inclination gradually changes to horizontal and it then forms a platform at a convenient height for loading on a wagon. The dull steel is collected and the sharp distributed by one of the surface teams. The slings of sharp steel are stood up in a row of

racks along the south side of the shafthouse, the racks being numbered to correspond to the working levels.

The accompanying illustration, Fig. 63, shows a plan of the drill shop. It will be largely self-explanatory after it is stated that the drills are transported from one part of the shop to another on flat-topped cars having rigid trucks which necessitate straight tracks and turntables. An extension of the track reaches four hand forges used in shanking the

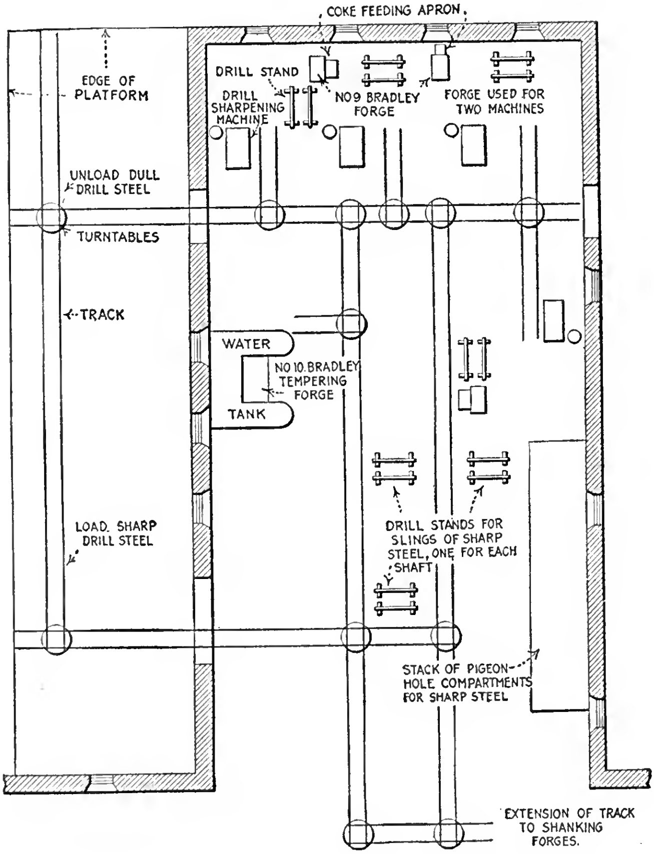


FIG. 63.—LAYOUT OF THE QUINCY DRILL-SHARPENING SHOP.

steel. The drill sharpeners used are a modified form of the Word machine. Drill stands consist of two wooden horses set at a convenient distance apart, their tops faced with a 3-in. strap of ½-in. iron. The tops of the cars are 36 × 48 in., and four iron stakes fit into tenons at each corner.

The sharp steel is sorted by boys in front of a stack of pigeon-hole compartments, drills belonging to each contract being kept in a separate

compartment. The pigeon holes are numbered in order by contracts, and the shaft and level numbers are also given. The sharpened drills are removed from the pigeon holes by other boys, who bundle them into slings; the slings going to each shaft are segregated on one drill stand, the shaft number being marked with white chalk on the ring, the contract number with blue chalk on the wedges. The slings are left on the drill stands until a truck load of them has accumulated; they are then wheeled out to the loading platform.

As compared with the practice of the other Lake mines, the Quincy system has the marked disadvantage that the drills are handled a greater number of times by hand than is usual; this disadvantage is largely offset by keeping the drills in bundles. The principal advantages of the method

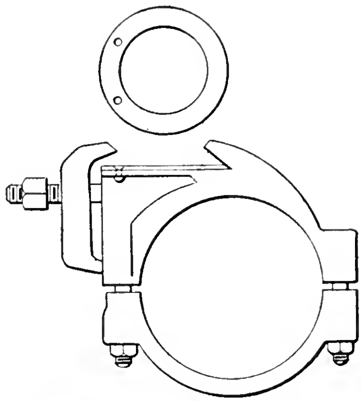


FIG. 64.—SHIM FOR WORN MACHINE-DRILL CLAMPS.

may be stated as follows: All sorting is done in the drill shop, where it may be easily supervised; drills for each contract are in rigid bundles, which tends to secure better distribution; the drill steel is easily handled while being distributed to the underground working places; this latter point is of especial force in minimizing delays to the hoisting plant, which is an important factor at these deep, low-grade mines.

Repairing Worn Clamp (By George E. Addy).—When the gripping portion of a machine-drill clamp or saddle becomes worn large, or when the saucer bases of the machines are worn small, it may be impossible to bring the clamp jaw to bear on the machine base. In such a case, the clamp may be continued in service by using the device illustrated in Fig. 64. The flat ring is fastened to the clamp, so as to act as a shim, and by slightly raising the base of the machine, enable it to be gripped by the jaw. In the drawing the ring is shown as riveted by countersunk rivets through the overhanging edge of the clamp. If this overhang does

not exist, countersunk screws can be substituted. They are better, except that they require more labor to put in.

Drill Tester for the Shop.—A device for obtaining graphically and rapidly many of the essential characteristics of rock drills without subjecting them to actual drilling tests has been devised by William D. Paynter, of Grass Valley, Calif., the machinist in charge of the repair and upkeep of machine drills of the North Star mine. Its general appearance with a stoper under test is shown in Fig. 65.

The base has a pedestal bolted to its left-hand end in which a vertical pillar is clamped by a split collar. This pillar carries a horizontal longitudinal arm on which may be clamped an ordinary saddle for a piston or Leyner machine, or a carrier for use as a back-stop with air-

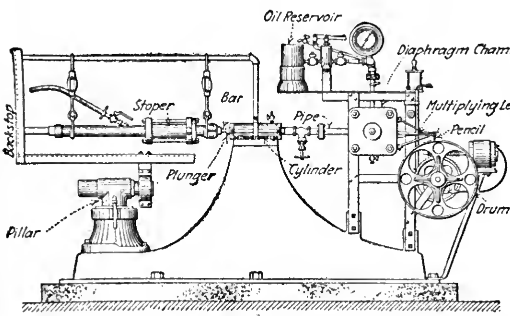


FIG. 65.—ELEVATION OF DRILL-TESTING MACHINE.

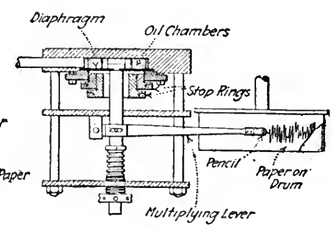


FIG. 65*a*.—PLAN OF DIAPHRAGM, RECORDING ARM AND DRUM.

feed hammer stopers. An air-feed stoper is shown suspended by hangers from a suitable support. The pillar permits vertical adjustment and the clamp gives a horizontal movement. The central part of the base rises to carry the cylinder and plunger device as shown. One end of the plunger projects through the cylinder and against this bears a bar, which is substituted for the bit in the drill. The other end of the cylinder communicates with a pipe to the diaphragm chamber, carried on a framework at the right-hand end of the machine.

The diaphragm chamber is shown in more detail in Fig. 65*a*. It consists of a plate, recessed to form a chamber which is closed with a diaphragm of highly tempered steel. This diaphragm is held under a cover ring and its motion is adjustable by studs on the inside and stop rings in the nature of bushings on the outside. The cylinder, pipe and diaphragm chamber are filled with oil by a pump from the reservoir shown, and there is also connected with this oil system a pressure gage. A rod is secured to the center of the diaphragm so as to move with it, and carries a spring which governs its motion somewhat. An arm called the

multiplying lever is moved by this diaphragm rod, being pivoted at one end and carrying a recording pencil on its long arm. The pencil bears against the paper on a revolving drum which is driven through worm gearing by a small motor.

The feed pressure and blows of the drill are communicated to the plunger by the bar which bears against the plunger end. They are

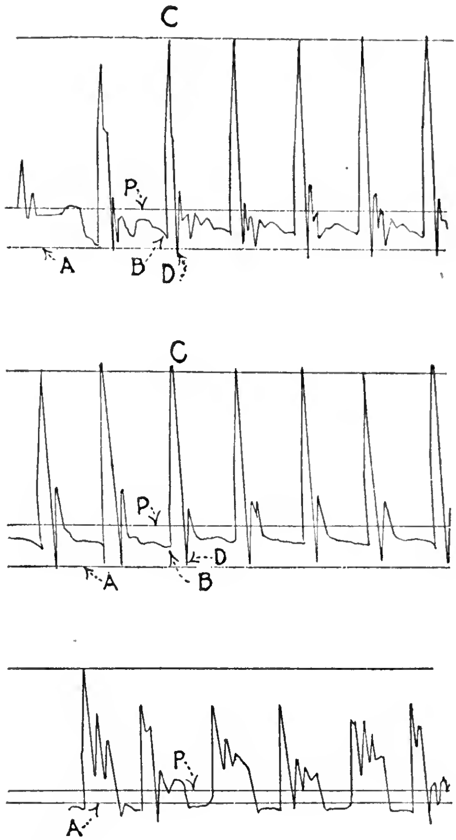


FIG. 66.—TYPICAL CARDS OBTAINED WITH THE TESTER.

transmitted by the non-compressible oil so as to vibrate the diaphragm, the movements of which are magnified by the pencil arm and recorded on the paper of the drum. The speed of the drum is uniform and can be calibrated by the use of a pendulum. The movement of the pencil across the paper, corresponding to the strength of the blows, can be calibrated by measuring it against readings on the pressure gage.

The method of testing is somewhat as follows: The drum with the paper on its periphery is revolved with no pressure on the diaphragm,

the line A, Fig. 66, is thus obtained. It is then revolved in the case of a stoper with the air pressure on the telescope feed, but with the hammer not moving, giving the line P. The drill is then allowed to reciprocate and its blows give the irregular line shown, the impact of the hammer forming the line BC and the return of the diaphragm, the line CD.

The upper card in Fig. 66 represents a test on a typical stoper in which the air pressure was 100 lb., the number of blows struck was 1284 per minute, and the foot-pounds developed per blow calculated to be 50. This would show a little less than 2 hp. developed. In the middle card, obtained from a similar machine, there will be noted a secondary blow, the cause and effect of which are not yet determined. In this case the air pressure was 96 lb., the blows per minute 1260 and the foot-pounds per blow, 40. The lower card was obtained from a machine in poor shape. The air pressure in this case was 84 lb., the blows per minute 1272, and the foot-pounds per blow, probably about 15. These last two mentioned machines drilling in hard rock under the same pressure, 84 lb., made 0.105 ft. and 0.051 ft. per minute, respectively, thus confirming the indications of the tester. It is evident that the tester fulfills much the same function as a steam-engine indicator in giving an energy graph, valuable, however, more for its indications of condition and adjustment than for the quantitative measurements. It differs from the steam-engine indicator in measuring output rather than input.

The development of the machine grew out of the necessity of preventing defective drills from going underground. At the North Star the machines are usually taken on a truck from the shop to the shaft collar, unloaded, and loaded on a cage, unloaded again at the shaft turn, loaded on a truck to go up the main raise, unloaded at a level station, loaded in a car and taken to the top or bottom of a stope, unloaded and carried to the working face. Evidently this is a process costing money. If the machine will not drill when set up, it means a definite loss to the company, augmented by the fact that the irritated miner will work less efficiently for the rest of the shift. It was found impossible to detect the defective machines in the shop. One which might sound all right and work well when set up and run against a block would prove quite useless underground. To get a more definite line on the conditions of the drills, a simple machine, embodying the principle described, was built and this more complete and convenient device was developed therefrom.

The tester has served this purpose excellently, but it also lends itself to investigation along other lines. The effects of changes in pressure, lubricant, etc., can all be investigated in this way more easily and more exactly than by actual drilling tests. A test to ascertain the effect of a difference in pressures indicated that a drop of 32 per cent. caused a

drop in the strength of the blow of 31 per cent., but a drop in speed of only 12 per cent. Underground tests showed that a drop in air pressure of 28 per cent. gave a decrease in drilling speed of 47 per cent. with sharp steel; a pressure drop of 22 per cent. gave a drilling-speed decrease of 46 per cent. with dull steel; a pressure drop of 28 per cent. in another case gave a drilling-speed drop of only 25 per cent.

A test with different lubricants showed that a change from heavy lubricant to medium gave an increase in blows per minute from 1116 to 1212 and of foot-pounds per blow from 25 to 34, while a change to light oil again gave increases to 1224 and 38. This test, however, was made on a new machine and it may be that smaller differences would be found in the case of an old machine.

The statement is frequently made that excessive air consumption indicates that a drill is in bad condition. A large number of tests indicated that machines of a certain type had an average air consumption of 75 to 80 cu. ft. of free air per minute at 90 lb. pressure. A test of a machine of this type using 90 cu. ft. gave 1296 blows and 43 ft.-lb., high figures, which indicate that increased air consumption does not necessarily indicate poor condition. A further field of usefulness lies in testing out new equipment. Thus in one instance four new valves were obtained from the manufacturers and when tested in machines, one was found defective. Investigation showed a piece of steel-cutting clogging a port so as to throttle the air and reduce the strength of the blow 35 per cent.

It would seem as if one of the most extensive fields of application of the device would be in comparing drills of different makes. Many inferior types could be eliminated at once in a preliminary test on the machine and only the high-class drills reserved for further test in rock.

Failure and Heat Treatment of Drill Steel (By Sven V. Bergh).—The steel generally used for rock drilling may be classified as carbon steel; hence the degree of hardness means the percentage of carbon contained. Experience has proved that the proper percentage of carbon is governed chiefly by the hardness of the rock to be drilled and by the power of the machine. The harder the rock, the softer must be the steel. This is due to the fact that the bit will not stand the impact if too hard a steel is used to penetrate hard ground, the degree of hardening necessary to improve the wearing qualities of the bit depending upon the conditions under which it has to be used. The above rule must also be applied when a heavier type of drill is being substituted for a lighter one and the steel does not seem to stand up well. The ultimate carbon hardness being already reached, a softer steel must be tried.

In some drilling practice, as, for instance, when hand-feed drills are used, bending stresses are likely to be put on the steel. Additional

tensile and compressive stresses are thus induced during the period of the blow, producing an excessive strain in the steel. In such practice a heavy size of steel is to be recommended. When the steel is shanked it is of importance to give the shank sufficient length and cross-sectional area. It happens not infrequently, where a light type of drill has been substituted for a heavier one, that the old steel is continued in use. This is not advisable, for the additional reason that it necessitates a consider-

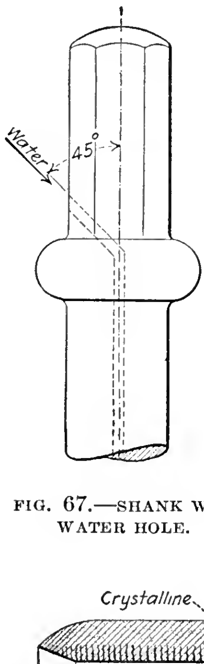


FIG. 67.—SHANK WITH WATER HOLE.

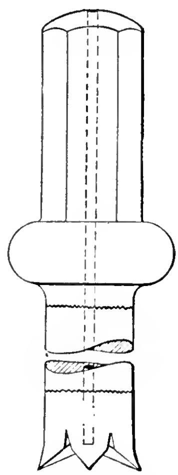


FIG. 68.—RUPTURE OF SHANK AND BIT.

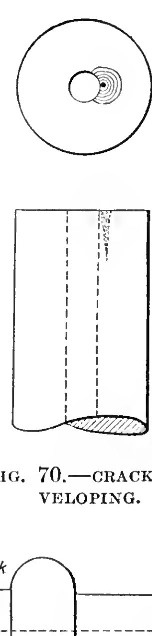


FIG. 70.—CRACK DEVELOPING.

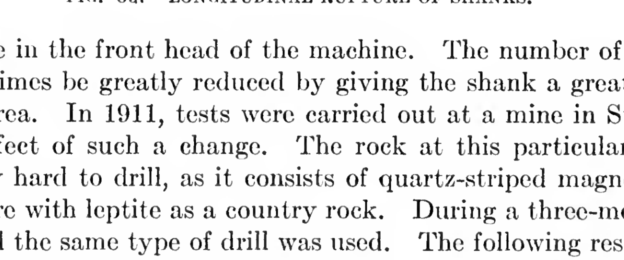


FIG. 69.—LONGITUDINAL RUPTURE OF SHANKS.

able change in the front head of the machine. The number of ruptures may sometimes be greatly reduced by giving the shank a greater cross-sectional area. In 1911, tests were carried out at a mine in Sweden to find the effect of such a change. The rock at this particular mine is exceedingly hard to drill, as it consists of quartz-striped magnetite and hematite ore with leptite as a country rock. During a three-month period one and the same type of drill was used. The following results were obtained: Ordinary shanks ruptured, 185, corresponding to 6277 m. drilled; strengthened shanks ruptured, four (171 used in all), corresponding to 700 m. drilled. These figures show a considerable improvement

following the use of the heavier shanks. Here it may also be mentioned that all abrupt changes in section and sharp corners should be strictly avoided.

So far as water-using drills are concerned, when the water is fed into the steel through a radial boring it has proved an advantage to drill the hole at an angle of 45° with the center hole of the steel, Fig. 67. To shape the shank and the bit, the steel has to be upset at both ends. This operation changes the internal structure of the steel unfavorably, at the same time producing in some parts of it the bad effect of "cold work," due to the manner of applying the heat. Ruptures as shown in Fig. 68 are frequently seen. Thus to utilize the best properties of the steel it is necessary that the upsetting be performed in such a way that the steel receives the proper heat treatment. From this it may be understood why it is regarded good practice to forge and harden the bit in separate heats. To obtain the finest grain, the bit has to be forged continuously from the highest temperature employed down to the finishing temperature, which probably is slightly above the point of recalescence. It is advantageous in hardening not to treat it to any higher temperature than necessary. The steel is liable, if heated too high, to change into a coarse crystallization and develop hardening cracks that may cause ruptures. The shank must always be tempered properly after being hardened.

Finally, it may be mentioned that shanks sometimes rupture longitudinally, Fig. 69. Some cases that were investigated showed that the failure was due to careless straightening after the steel had been upset. The center hole of the shank was thus oval instead of round and this later caused it to rupture when the steel was used.

Two kinds of rupture may be distinguished, namely: (1) Ruptures that appear as developed from a coarse-grained structure; (2) ruptures that evidently have developed from a partial break or inclosure in the steel. In regard to ruptures of the first class mentioned, the opinion frequently heard among users of drill steel is that the crystalline texture often found is due to vibrations. It is not the intention here to discuss whether such an opinion is wrong or not, but to call attention to the fact that the crystalline texture might have been caused during the manufacture of the steel bars by wrong or insufficient heat treatment. The crystallized steel is sometimes hard to restore completely. By annealing, the crystals may only be more or less broken up, still maintaining their previous orientation. Thus cleavage planes are developed along which ruptures occur, giving the appearance of a crystalline fracture, although the steel in reality is fine-grained.

The typical fracture of the second class is as follows: A circular or oblong cavity or a crack is found at some part of a transverse section. This cavity or crack is surrounded by concentric rings covering a certain

portion of the section, the rest being more or less covered by an ordinary crystalline fracture, Fig. 70. It has been stated that these inclosures seemed to run through the whole length of the steel and if it was broken anywhere, one could expect to find them. It is a fact that this kind of rupture occurs rather often. It is of practical importance to determine whether the inclosures mentioned consist of slag or of sulphide of manganese, the latter usually occurring in the form of round drops, which, if large in size, may be elongated by the rolling. Sulphide of manganese is best recognized by etching the surface of the fracture with a mixture of dilute hydrochloric acid and bichloride of mercury, when the sulphur appears as dark spots. A print may also be taken conveniently by exposing the fracture for 4 or 5 min. to a piece of silk wet with the solution.

It may be of interest to many to hear that ruptures have also been found to start from groove marks on the steel, where numerals or other marks have been stamped. It is evident from what is now said that the annealing of the drill steel at intervals will mean only a partial improvement. Internal strains produced by cold working of the metal and coarse-grained crystals may be effaced, but there is no possibility of eliminating ruptures of the second class noted. This is in accord with what has been found at various Swedish mining fields where the practice is to anneal drills every second month. The structure of steel exposed to vibrations has proved to be of utmost importance. The sorbitic structure has been found to stand up best. It is obtained either by cooling the steel quickly through the so-called critical range without actual quenching or by rapid cooling and then reheating to about 600° C.

IV

SHAFTS AND RAISES

Sinking and Timbering—Concrete Shaft Lining—Concrete Skip Stringers —Stations—Raising—Ladders

SINKING AND TIMBERING

Shaft Timbering in Minnesota Iron Mines (By L. D. Davenport).—In Fig. 71 is shown the timbering for a shaft, 6 × 18 ft. inside, sunk through surface, sand and ore in the Chisholm district of Minnesota. No water was encountered until the ore was reached and then only a small quantity. After the shaft was staked out, two level trenches were dug on the surface and two bearing pieces or carriers, 40 or 50 ft. long (2- or 5-ft. flattened timber), were placed in them about 10 ft. apart. These 40-ft. bearing pieces were parallel to the wall plates and when they were leveled up, two cross-carriers 12 × 12 in. by 20 ft. were placed at right angles across them 18 ft. apart. The first set was then put in place with the end pieces, 12 × 12 in. by 8 ft., resting on the cross-carriers.

The next set was then put in place below the carriers and the studdles of this set fitted into ½-in. joggles, cut in the cross-carriers. These studdles were cut off on account of the cross-carriers, so that the distance between the sets was 3 ft. 11 in. The 12 × 12-in. studdles in the regular sets were 4 ft. long.

The first set below the carriers was like the other sets; it was held by four hanging bolts of 1¼-in. round iron. Next, the set which forms the collar was placed above the set resting on the carriers, and diagonally braced to both sets of bearers. Four straight hanging bolts were used here instead of the hooked bolts used in the other sets. The bolts for this collar set were made 1 ft. longer than the regular 6 ft. 8 in. length, on account of their passing through the cross-carriers as well as the end plates. As the dirt was hoisted it was banked around the shaft collar, and served not only to steady the timber, but also to keep out the surface drainage.

The middle compartment, used for hoisting, was boarded up as fast as the dividers were put in, so that the bucket would not catch. The dividers were put in as the shaft was sunk, but the last two sets were always left open to allow the wall plates of the next set to be handled.

When the end plates and wall plates were framed, a strip 2 × 2 in. was spiked on the outside of each piece in the center, to make a bearing for the lagging. The lagging was 2-in. plank about 4 ft. 9 in. long. Four hanging bolts to each set were usually enough, but in heavy ground one or two more were sometimes used in the wall plates. Divider studdles or center studdles, 6 × 12 in. by 3 ft. 11 in., were put in after

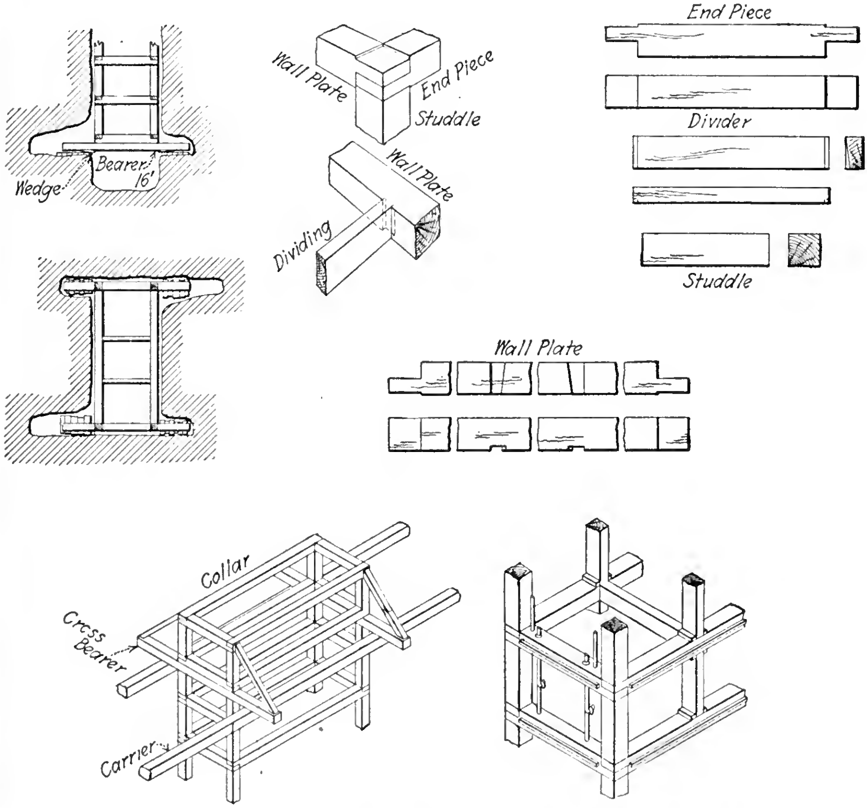


FIG. 71.—DETAILS OF SHAFT TIMBERING IN THE CHISHOLM IRON MINES.

the dividers, and since these studdles were not gained in to the wall plates, small cleats were used to hold them in place.

The second set of carriers were 12 × 12-in. timber 16 ft. long, and to put them in place, small drifts, about 3 × 4 ft., were run in for approximately 8 ft. at the end of the end plates at two diagonally opposite corners of the shaft. Smaller drifts were run in about 4 ft. at the other two corners. The inside corners of the 8-ft. drifts were rounded off, the bottoms trued up, and short pieces of plank laid crossways in them. A bearer was then brought down and one end run into the 8-ft. drift.

Next it was swung parallel to the end plate and the other end run back into the 4-ft. drift. The same thing was done with the other bearer and the two rested on the bottom boards of the drifts, directly under the corresponding end plates. Long, flat wedges were then driven under the bearers on top of the boards and as they were raised against the end pieces, the hanging bolts were tightened to take up the weight. When the bearers were up tight against the set and well wedged, the small drifts were carefully blocked up and the lagging put in.

Cage and Bucket for Sinking (By Claude T. Rice).—The No. 2 vertical shaft of the Hancock Consolidated Mining Co., in the Lake Superior copper region, was sunk to intersect the Pewabic amygdaloid at a depth of 3500 ft. Although primarily an exploration shaft, the dimensions, $9\frac{1}{2} \times 29\frac{1}{2}$ ft., are those of a working shaft. It is the second largest shaft in the copper country, with four hoisting compartments, each 7 ft. long by 5 ft. 2 in. wide, and one 4×7-ft. ladder- and pipe-way.

The ground excavated in sinking to a depth of 2600 ft. was raised to the surface in buckets. Then a change was made and at the 34th level a station was cut and bins built. The material was thereafter raised in buckets from the bottom of the shaft to the bins and thence was discharged into large skips in which it was raised to the surface. A rock pentice was left below the skip compartments and to prevent anything falling down the shaft upon the miners' platform, covers were used over the compartments in which the buckets ran, both at the 34th level and, during the first stage of the sinking, at the surface.

The bucket was hung from a cage, instead of a crosshead, as shown in Fig. 72. Two chains were fastened to the clevis of the hoisting-rope socket, and on the end of each a pair of sister hooks were fastened, which hooked into ears on the bucket. At first the hooks were fastened together by a pin, scissors-fashion, but were later used as satisfactorily without it. A cradle-car was used for handling the bucket on the surface. On the bottom of the bucket were two lugs, placed so that the pin, which was put through the lugs in attaching the bucket to the cradle when dumping, would be parallel to the line of the eyes of the bucket ears.

In the bottom of the cage a hole 3 ft. 7 in. in diameter was cut, the diameter of the bucket at the top being 3 ft. 2 in. A flanged ring held the bucket in position while being hoisted. There was room between the bucket and the rim of the hole to permit the heel of a man's shoe to pass up through as he rode the bucket from the shaft bottom up to the cage at the beginning of a hoisting trip, the cage hanging in the shaft on bumpers fastened to the guides at the bottom set of timbers. The hoisting rope passed freely through a hole in the center of the top piece of the cage, which was made without a hood in order to facilitate the lowering of timbers.

The shock of picking up the cage was absorbed by a rubber buffer on the hoisting rope, the buffer being about 9 in. in diameter and 6 in. thick and carried on an iron plate of the same diameter and 3/8 in. thick, which rested directly on the top face of the rope socket. The bucket, when in the cage, hung about 5 in. below the deck, but entered the flange far enough to be steadied. Friction between a maple filler in the top piece of the cage and the rubber buffer, prevented spinning. The deck of the cage made a fairly tight cover over the hoisting compartment when the cage rested on the bumpers at the bottom set of timbers.

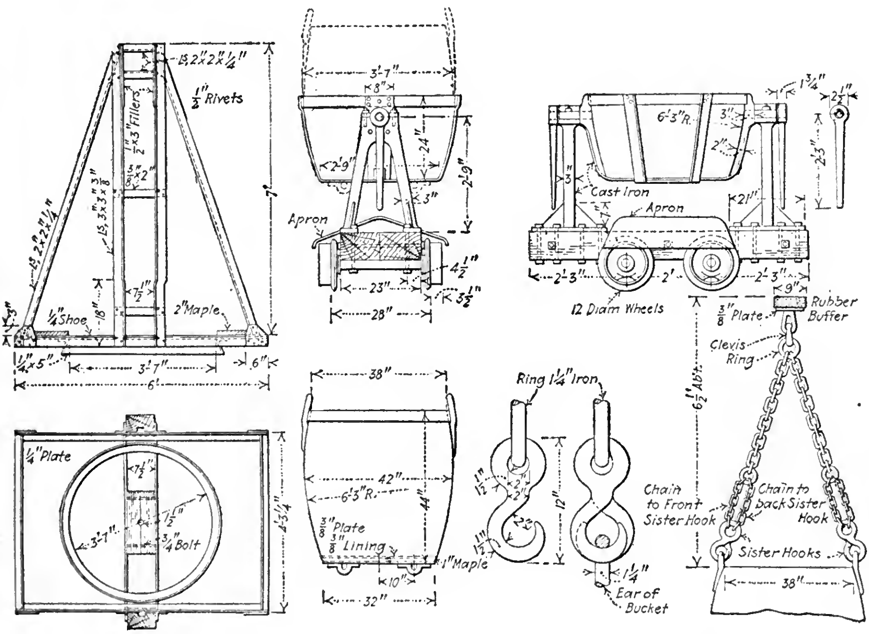


FIG. 72.—DETAILS OF HANCOCK BUCKET CAGE AND LANDING CAR.

At the surface each bucket compartment was covered by a platform that traveled along the guides. As the cage dropped below the collar of the shaft, the corresponding compartment was tightly covered. The cross-timbers of the platform extended out so as to catch on the top set of timbers at the collar of the shaft. Extending down from the frame proper of the cover were two timbers and the top piece of the cage, as it came up, struck these and carried the cover up the headframe ahead of it.

When the bucket was detached from the hoisting cable, the shaft compartment was covered by a sliding platform or landing carriage that was pushed by compressed air across the compartment through grooves in the guides. On this carriage were tracks for the cradle-car. In each bucket compartment and at some distance above the collar of the shaft

was a gate timber that could be pulled under the cage as a chair to hold the cage while the bucket was being lowered to the cradle car on the landing carriage. The gate timber was fastened to a counterweight so that, as soon as the cage was lifted, it would move out of the way of the cage.

In making the change of buckets at the surface the car that carried the last bucket sent down on that side would be left standing in such position on the landing carriage that when the carriage was moved into the shaft the car would come directly under the bucket ready to receive it. On the other side of the shaft was standing the car with the bucket in its cradle that had just been dumped. As soon as the cage was resting on the gate timber, the landing carriage was run in and the loaded bucket landed in the empty cradle. This car was given a push to the end of the carriage, the car with the empty bucket was run on and that bucket attached to the rope. Then, as soon as the bucket was raised

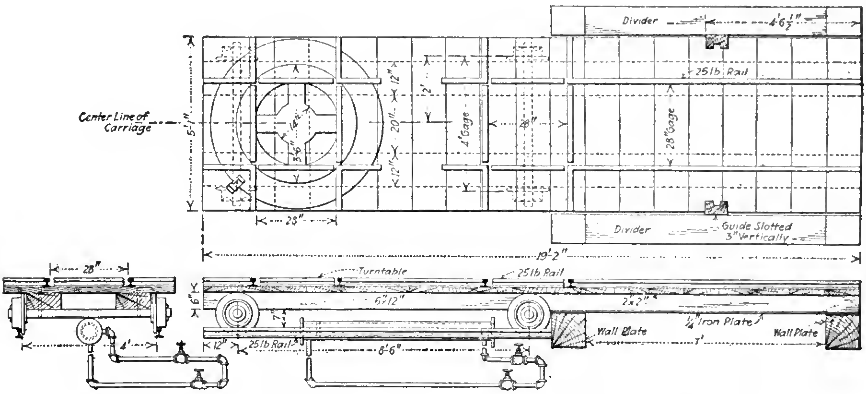


FIG. 73.—BUCKET-LANDING CARRIAGE USED AT TRANSFER STATION.

out of the cradle and had picked up the cage, the carriage was moved out of the way.

At the 34th level the buckets were attached and detached from the cable on the same side of the shaft. Fig. 73 shows the landing carriage used. This carriage was moved in and out of the shaft by an air cylinder. The carriage was thrown in so that the shaft would be completely covered while the bucket was being landed, and a cradle-car was run in under the bucket. In order to allow the carriage completely to cover the shaft compartment the guides were cut in two at the station and a 3-in. slot was left for the frame planks to slide through as the carriage moved across the shaft; thus no opening was left around the shaft at any time.

Having seen the moving of the bell cord as the men in the shaft signaled to the hoisting engineer to take the bucket up, the three landers, on the 34th level, would open the doors of the compartment and get ready to

land the bucket. When the cage had cleared the top of the station, a gate timber, similar to the one used at the surface, was pulled in across the shaft so as to catch the cage. After the bucket had been landed in the cradle, at the same time being turned so that the ears would be in line with the long axis of the car and out of the way when dumping, the sister hooks used to fasten the bucket to the hoisting rope were taken off, and the bucket was trammed over to the bin pockets by way of a turntable which was brought in line with the station track as soon as the landing carriage was moved into position in the shaft. While the bucket was being trammed to the bin, one of the three landers would stick a short piece of round iron through the lugs on the bottom of the bucket; this rod was long enough to extend past the straps forming the bottom of the

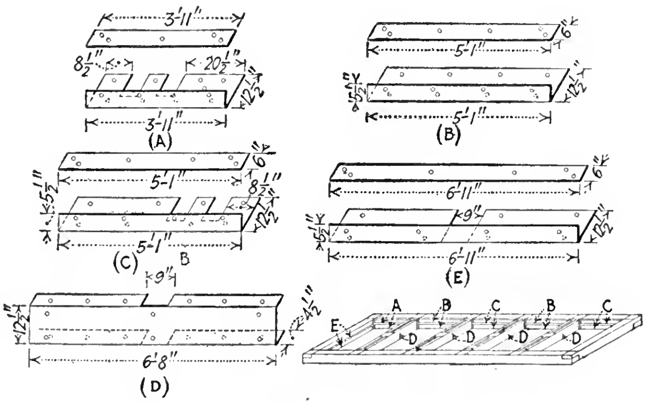


FIG. 74.—BLASTING IRONS USED IN HANCOCK SHAFT.

cradle; thus the bucket was securely held in the cradle while being dumped.

Kimberley skips were used to take the muck from the 34th level to the surface. These had a capacity of 8 tons and weighed about 5 tons empty. The maximum hoisting speed was 3500 ft. per minute with a total load of 20 tons, including the rope. The speed of hoisting with the bucket below the 34th level was 2500 ft. per minute.

Blasting Irons for Shaft Sets.—In sinking the No. 2 Hancock shaft no cover platform was used for storing the machine drills and protecting the lower set of timbers from flying rocks at blasting time. All the drilling equipment was raised to the 34th-level station at blasting time. Blasting irons, the details of which are shown in Fig. 74, were used to protect the bottom timbers from being cut by flying rock. These were made of ¼-in. iron plates, punched with ¾-in. holes, so that they could be fastened to the timbers with railroad spikes. Set *A* was for the wall plate in the manway compartment, being cut away for the hanging bolts on

the under side. One piece of the manway iron went under the wall plate and half way up the side, while to cover the other half of the inside space a separate piece was spiked on. All the other irons for the wall plates were similarly designed, but two other sets were needed, one pattern for the two compartments in which there were no hanger bolts, and the other for the two compartments in which the hanger bolts were used. The blasting irons for the end plates were similar to those for the wall plates, except that they were cut away on the under side to straddle the guides and studdles. The irons for the dividers were made in halves, each of which covered one side and half the top and bottom. The halves were put on the timbers and then held together by two clamps, also cut away to straddle the guides and studdles. The divider blasting irons could

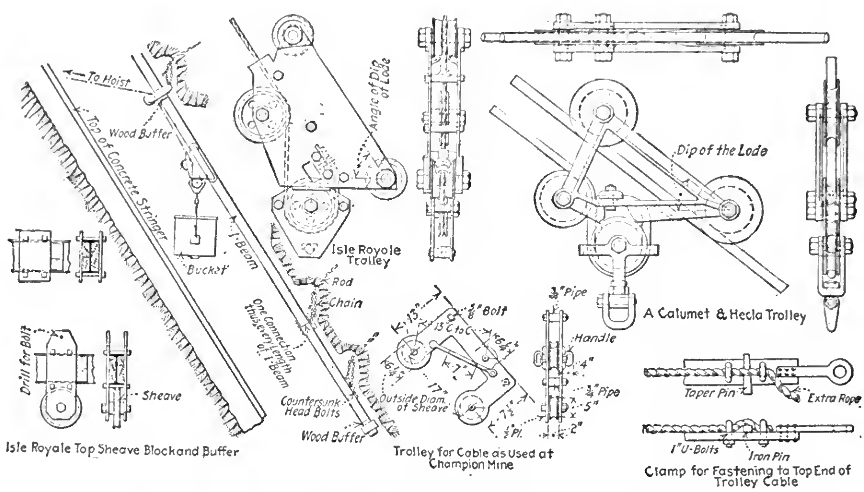


FIG. 75.—TROLLEYS USED IN SINKING INCLINE SHAFTS IN LAKE SUPERIOR COPPER COUNTRY.

be fastened on by spikes like the others. Spiking was preferred for the irons on the end and wall plates, owing to the blocking, and because when the ground was bad they could not conveniently be attached by clamps. The several patterns and their place on the set are shown in the illustration.

Trolleys for Sinking Incline Shafts.—Buckets are used in sinking incline shafts at the Lake Superior copper mines. These are carried by a trolley that runs either on a wire rope or on an 8-in. I-beam suspended from the roof by chains, as shown in Fig. 75. In order to keep the I-beam from swinging upward with the pull as the bucket is being loaded, it is usual to hang the top length of the I-beam from rods anchored in plugs in the roof, and to butt the upper end against a timber. This top length is curved, to allow the bucket to be carried high enough for dumping

into a car. Because of the fact that the I-beam does not sag as does a rope, larger buckets can be used with the I-beam equipment. This is a great advantage, while it is also possible to continue sinking as far as desired; with a rope trolley 200 ft. is as far as can be sunk at one lift; on the other hand the rope method may be cheaper.

The I-beams are fastened together by top and bottom plates, so as not to interfere with the travel of the trolley against the webs of the beam. Beams 20 ft. long are used, but an extension piece 10 ft. long is put in until the shaft can be worked deep enough to allow the use of a regular length of track. To prevent overrunning, a wooden buffer is bolted to the I-beam. In order to get a binding action in the drill holes that carry the I-beams, the holes are put in at right angles to the dip of the lode. These are drilled by the machines as the shaft is being sunk.

In shafts where a rope trolley is used, the bottom end of the rope is fastened to an eye-bolt wedged into a hole about 2 ft. deep which points back into the shaft so that the harder the pull the more the bolt is forced into the hole. At the upper end provision must be made for stretching the rope so that there will be as little sag as possible, for there can be no supporting of the trolley cable from intermediate points. In the South Hecla shaft of the Calumet & Hecla it is the custom to do the final tightening of the trolley cable by a turnbuckle having a travel of several feet. This is seated at one end against a timber running from the foot to the hanging wall, while the other is fastened by a clip to the rope. The extra trolley rope is carried in a coil out of the way. As new cable is required by the deepening of the shaft, the clip is loosened and more rope is let out. Then the clip is bolted on again and after pulling the cable as tight as possible with the hoisting engine, the turnbuckle is used to do the final stretching.

At the Mohawk mines the upper end of the trolley cable is bolted to a plate which is fastened to a bolt through a timber of the shaft. By tightening the nut on this bolt, the final stretching of the trolley cable is effected. The manner of fastening this clamp to the rope is shown in the illustration. There are two screw-bolts with an iron pin between them to put a kink in the cable and tighten it after the two U-bolts have been clamped upon it. Even when tightening as much as is practicable on the trolley cable, there is some sag, and even if the cable can be carried nearer the roof than the I-beam, it is necessary to use a smaller bucket with the rope than with the I-beam. At the South Hecla the sinking bucket has a capacity of 700 lb. of rock.

Whatever the method of carrying the trolley, the bucket is permanently attached to the hoisting cable by a fall that permits it to be lowered to the rock pile in the shaft. The illustration shows the rope trolley used at the Calumet & Hecla mines. The air hoist may be installed

either in the manway compartment of the shaft or in the stations, according to the preference of the superintendent in charge of the work. But installing the hoist in the shaft does away with the necessity of placing a sheave at the top around which the hoisting cable is passed.

Hinge for Shaft Doors (By Clinton P. Bernard).—While sinking is in progress it is usually necessary to place doors over the shaft at the collar for preventing muck from going down, and for carrying the rails if the material be dumped into a car. Fig. 76 shows a hinge for such doors; it can be made by any blacksmith, and has several advantages over the ordinary flat hinge. The center of the eye is set level with the rail and away from the edge of the shaft a distance equal to the thickness of the

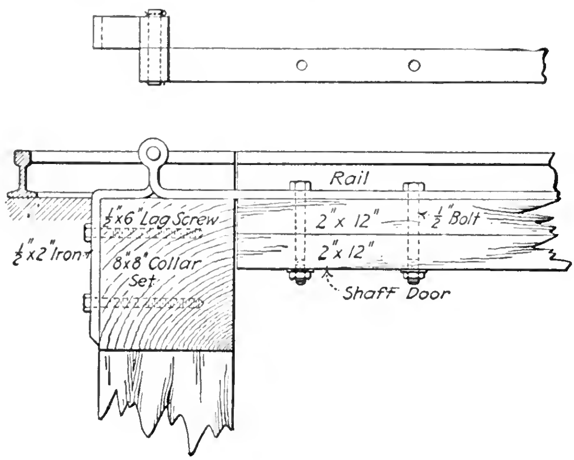


FIG. 76.—SHAFT DOOR WITH WROUGHT HINGE OF SPECIAL DESIGN.

door plus the rail. The hinge allows a close joint between the rails, while the door in a vertical position is entirely clear of the shaft, and when thrown back the rails are superimposed, thus relieving the hinge of all strain. The door half of the hinge has a bearing on the collar set to give added stiffness to the door.

Sinking an Untimbered Shaft.—The method of sinking shaft No. 6 of the St. Louis Smelting & Refining Co. in Missouri was novel in that it was carried down through limestone for 557 ft. without timbering and by means of a derrick instead of a headframe. Air pipe was not installed in permanent form until the completion of the shaft. No pumping was necessary, and only some slight timbering, where the limestone was badly shattered. A speed of 100 ft. per month was maintained. After the bottom was reached, timbering was done, working from the bottom up. Electric lights were strung through the shaft while the permanent timbers and pipe were being set. The end of the derrick boom also

carried a cluster of electric lights for illuminating the ground around the collar of the shaft. Mucking was done by means of a cluster of electric lamps on the end of a wire raised and lowered by a reel at the collar. To provide against failure of the current, two torches were always kept tied to the bucket.

Hanging Bracket for Shaft Platform.—In sinking the Bennett shaft on the Mesabi range, the device shown in Fig. 77 was used to support a hanging platform, in cases where the ground could be left open for some

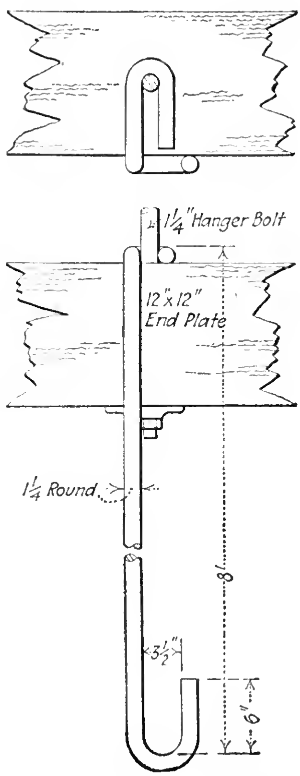


FIG. 77.—BRACKET SUSPENDED FROM HANGER BOLTS.

distance below the timbering and it was therefore impossible to swing a set from the bottom. Four pieces of 1¼-in. round iron, the same that was used for the hanger bolts, were bent at one end so as to hook around the hanger bolts supporting the lowest end plate and then drop vertically over the side of the end plate. The other ends were bent to form loops of the proper size to take a 3-in. pipe, and in such a plane that the pipe when inserted would lie parallel to the wall plates. On the two lengths of pipe inserted in the four brackets, a plank platform was built on

which the men could work in placing the new set. The sets were spaced 5 ft. and the length of the bracket, 8 ft. overall, was sufficient to give plenty of clearance below the new set and room to work.

Aligning Concrete Forms in Shaft (By Robert H. Dickson).—The Calumet & Arizona Mining Co., of Bisbee, in relining its Junction shaft started at the bottom and replaced the timbers with concrete. Since most of the shaft sets had been in place for 10 years, and in that time moved more or less from their original position, permanent points had to be established at regular intervals in the shaft, for lining in the concrete forms.

A preliminary plumbing was done in order to ascertain the relative positions of the sets, and thus determine the best position for the perma-

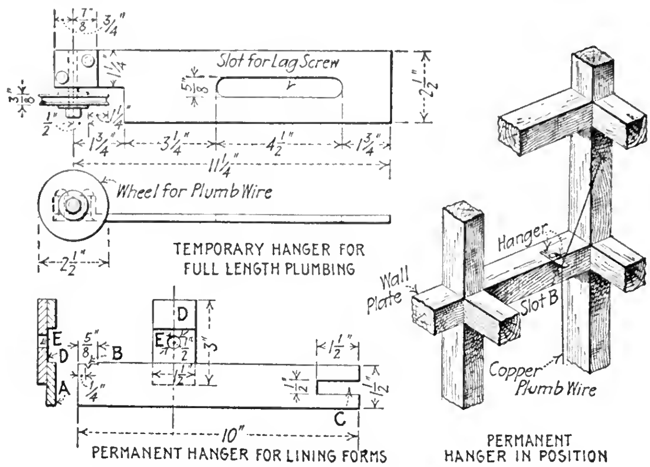


FIG. 78.—HANGERS FOR SHAFT PLUMB WIRES.

nent points. This plumbing was done by hanging two wires in opposite corners of the shaft, and measuring the distances from them to the wall plates and end plates.

The permanent points were placed in proper position by hanging two plumb lines the full length of the shaft. The lines were manipulated over hangers, shown in Fig. 78. These hangers were first set by hanging a short plumb line over them, and adjusting this into proper position by transit and tape. They were held by a 5-in. lagscrew and washer to a 10 × 10-in. stringer, projecting a little beyond the edge of the shaft. The ⅝-in. slot, through which the lagscrew passed, allowed the hanger to be moved transversely or longitudinally, and thus adjusted to exact position. The long plumb lines of piano wire were then hung over the hangers. Each line was 1500 ft. long and carried a 60-lb. weight partly immersed in crude oil.

There was a strong upward draft in the shaft, due to the hot steam pipes supplying the pumps, which caused troublesome vibration of the wires. An ingenious scheme was devised by E. E. Whitely, chief engineer of the company, to minimize this vibration. It consisted of finding the center of swing of the wire at intervals of 300 ft. and then fastening it in this position. A strip of wood about $\frac{1}{4} \times 1 \times 12$ in., with a $\frac{3}{16}$-in. hole near one end, slotted to insert the wire, was set so that the wire vibrated in the hole without touching the sides. When the center of vibration was at the center of the hole, the strip was nailed to the wall plate, and two halves of a pencil about 2 in. long, with the lead removed, were placed in the hole so that the wire passed through the groove left by the head, thus holding the wire fast in place.

The next step consisted in placing permanent points or hangers at 35-ft. intervals. Their construction is also shown. They were made of $\frac{1}{4}$-in. iron plate sheared into strips A, $1\frac{1}{2} \times 10$ in. One end of one edge of the strip was milled for a distance of $\frac{5}{8}$ in. back and in this a file-cut B was made so as to be vertical when the hanger was in place. The other end of the strip was slotted at C. Two of these were placed at each 35-ft. interval in opposite ends of the shaft but near the same wall plate. They were fastened to the wall plate by lagscrews through the slot C so that the wire passed through the little slot B. By means of the washer D with a screw through the $\frac{1}{2}$-in. hole E, the hanger was held firmly to the timber so as not to rotate about the screw through C. For actually lining in the concrete forms, a copper wire with a 10-lb. plumb bob was fastened to a post or other timber, as shown, and then let down through the slot B. As the work proceeded upward, the hangers were taken out with the timbers and the next set above used.

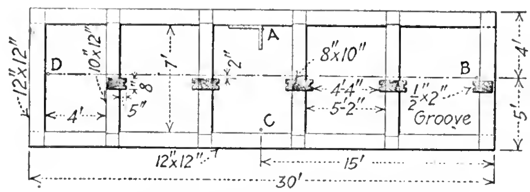
Setting Timbers in Vertical Shafts (By C. W. Macdougall).—The following method of setting timber in a vertical shaft was used during practically the entire sinking period of one of the largest shafts in the Michigan copper country, and was found thoroughly reliable. It possesses certain advantages over the usual method.

The shaft in question is about 4000 ft. deep with a rock section of about 10×31 ft., or 9×30 ft. outside the timbers, and has four hoisting compartments besides a ladderway. The long axis of the shaft lies about north and south. All plates are 12×12 in. and the dividers 10×12 in.; the hoisting compartments are 5 ft. 2 in. by 7 ft. overall and 4 ft. 4 in. between the guides. The guides are 5×8 in., with a groove 2 in. wide and $1\frac{1}{2}$ in. deep down the center. The framing of all the timber was done from templates.

A vertical saw-mark was made on the inside of the wall and end plates, that on the wall plate being in the center of the timber, and that on the

end plate being 4 ft. from the foot-wall end, so that the lines would clear the guides.

The sets were hung and temporarily blocked approximately in place in succession, and when all that were required had been assembled, plumb lines were dropped as shown in Fig. 79 at points *A*, *B*, *C*, *D*, from reference points in the sets previously placed above. Each set was then shifted by means of wedges at the four corners until the saw marks on the timbers were in line with the plumb lines, this being determined by means of an ordinary carpenter's square, held as shown at *A*. After the four corners were blocked into place, a line was stretched the full length of the shaft from *B* to *D*, just above the tops of the dividers. The set was then blocked at points opposite the ends of the dividers until the east side of the guide studdle on each divider was 2 in. from the line. The object of having the lines stretched from *B* to *D* and taking measurements on each divider was to keep the wall plates from bending in the middle, as they would if the wedges were not driven evenly on both sides of the shaft.



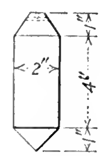


FIG. 79.—METHOD EMPLOYED FOR LINING UP SHAFT TIMBERS.

By using this method, the distance the plumb lines were hung from the inside face of the timbers was immaterial, the practice in this case being to hang the lines from 60-d. nails about 3 in. from the timber and to use 4-lb. plumb bobs, of the design shown, for weights, the plumb lines being of No. 30 trot line, corresponding to a heavy mason's line. The reference points by which the plumb lines were set were copper tacks with their heads flattened until parallel with the long axis of the shank, these points being marked by aluminum tags nailed to the timbers just over them. This method of lining up the timbers proved to be far quicker than the method of hanging the lines in the four corners of the shaft and using wooden blocks for gages, for the wooden blocks would invariably touch a line, set it swinging and make it necessary to steady it again, costing considerable time; whereas, in the method shown, the lines were seldom disturbed and there was far less danger of error.

Light Shaft Timbering (By Harold A. Linke).—A convenient size for a vertical, two-compartment, exploratory shaft is 3 ft. 6 in. by 7 ft. 8 in. inside timbers. The excavation for a shaft of this size measures

about 5 × 9 ft., is of minimum cross-section for convenience and still large enough to permit drilling to sufficient depth for economical work.

The problem of timbering a shaft is governed by local conditions. In case the ground is not heavy and stands well, where only occasional lagging is required and where the shaft sets serve mainly as support for the guides, ladders, pipe, etc., the sets shown in Fig. 80 at 1 answer every purpose. The joint is detailed in 2. The ends of the posts for this set require no special framing other than squaring. If, however, all available strength of set timbers is required, then it is advisable to frame as shown either in 3 or in 4. The end of a post for a set framed in this manner is detailed in 5. In 6 are shown the details of a divider.

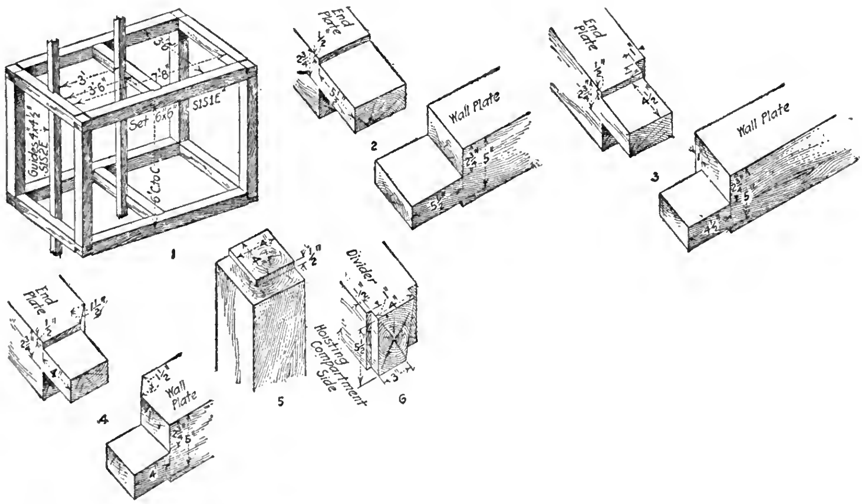


FIG. 80.—SHAFT SET OF LIGHT TIMBER AND DETAILS OF JOINTS.

When lumber is to be transported by rail for any considerable distance it will be found that the laid-down price of, say, 6 × 6-in., S1S1E is much lower than 6 × 6-in. rough; that is to say, the saving in transportation on weight removed in surfacing more than compensates for the cost of dressing. The details 2 to 6 are dimensioned for 6 × 6-in. S1S1E, which usually measures 5½ to 5⅝ in. square.

Diagonal End Plates, Inclined Shaft (By G. A. Denny).—For a Mexican shaft, inclined at 65°, there was used in the first stages the usual type of timbering. The hanging in the shaft, however, was found exceedingly heavy and hard to support, and to overcome this difficulty the system of timbering shown in Fig. 81 was designed. This proved to be altogether superior to the first timbering and easily sustained the pressure put upon it by the heavy ground.

The principal difference between the two systems of timbering is, of

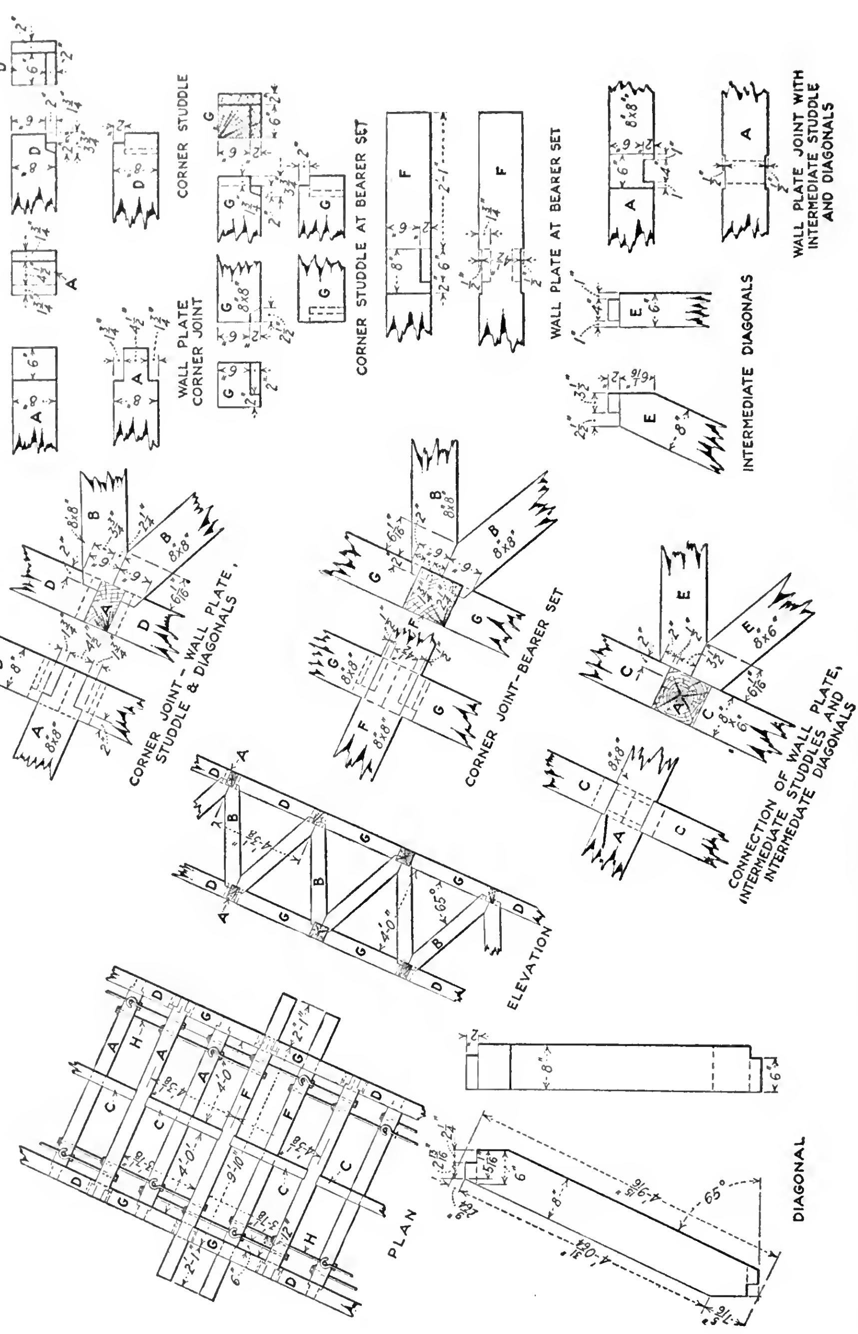


FIG. 81.—THE NEW TIMBERING WITH DIAGONAL MEMBERS.

course, the substitution of the diagonal end plates for the right-angled. The bearers, however, are put in parallel to the wall plates in the new system and parallel to the end plates in the old.

TABLE I.—TIMBER LIST OF COLLAR, ORDINARY AND BEARER SETS OF OLD SYSTEM

No. per set	Size	Length	Material	Description
2	12 × 12″	20′ 0″	Pine	Wall plates, collar set.
2	12 × 12″	4′ 6″	Pine	End plates, collar set.
2	6 × 12″	4′ 6″	Pine	Dividers, collar set.
4	1¼″ diam.	6′ 2⅞″	Wrt. iron	Tie bolts, collar set.
8	1″ diam.	3′ 2″	Wrt. iron	Hanging bolts, collar set.
2	8 × 8″	14′ 6″	Pine	Wall plates, shaft set.
2	8 × 8″	5′ 6″	Pine	End plates, shaft set.
2	6 × 10″	4′ 6″	Pine	Dividers, shaft set.
4	8 × 8″	4′ 6″	Pine	Corner studdles, shaft set.
4	4 × 10″	4′ 6″	Pine	Intermediate studdles, shaft set
8	1″ diam.	2′ 10″	Wrt. iron	Hanging bolts, shaft set.
2	8 × 8″	14′ 6″	Pine	Wall plates, bearer set.
2	8 × 8″	5′ 6″	Pine	End plates, bearer set.
2	8 × 8″	8′ 6″	Pine	Bearers, bearer set.
2	6 × 10″	4′ 6″	Pine	Dividers, bearer set.
4	8 × 8″	4′ 0″	Pine	Corner studdles, bearer set.
4	4 × 10″	4′ 6″	Pine	Intermediate studdles, bearer set.
8	1″ diam.	2′ 10″	Wrt. iron	Hanging bolts, bearer set.

TABLE II.—TIMBER LIST OF ORDINARY AND BEARER SETS IN NEW SYSTEM

No. per set	Mark	Size	Length	Material	Description
2	A	8 × 8″	9′ 10″	Pine	Wall plates, shaft set.
4	B	8 × 8″	4′ 10⁵⁄₃₂″	Pine	End diagonals, shaft set.
2	E	6 × 8″	4′ 10⁵⁄₃₂″	Pine	Intermediate diagonals, shaft set.
4	D	8 × 8″	3′ 10⅝″	Pine	Corner studdles, shaft set.
2	C	6 × 8″	3′ 8⅛″	Pine	Intermediate studdles, shaft set.
8	H	1″ diam.	2′ 5⁹⁄₁₆″	Wrt. iron	Hanging bolts, shaft set.
2	F	8 × 8″	14′ 0″	Pine	Wall plates, bearer set.
4	B	8 × 8″	4′ 10⁵⁄₃₂″	Pine	End diagonals, bearer set.
2	E	6 × 8″	4′ 10⁵⁄₃₂″	Pine	Intermediate diagonals, bearer set.
8	G	8 × 8″	3′ 9⅜″	Pine	Corner studdles, bearer set.
2	C	6 × 8″	3′ 8⅛″	Pine	Intermediate studdles, bearer set.
8	H	1″ diam.	2′ 5⁹⁄₁₆″	Wrt. iron	Hanging bolts, bearer set.

Table I gives the bill of materials for sets of the first system and Table II gives it for the new system.

Manway and Skipway Door.—The Bennett shaft near Keewatin on the Mesabi range is divided into a combination man- and pipeway and two skipways, arranged in a row. A vertical partition or brattice of 2-in. planks separates the manway from the adjacent skipway. The shaft sets are spaced 5 ft. and on every third set a sollar is built of 2-in. planks laid on 2 × 6-in. joists parallel to the end plates. Repair work on the pipes necessitates the lowering of material and tools, which or-

dinarily must be carried down by hand or slung from sollar to sollar. To enable such supplies to be lowered in the skip and taken into the manway, a door has been provided in the brattice at each sollar. The ladder opening in the sollar is 2 ft. square, situated in a corner next to the skipway. The door occupies the place of the second and third brattice boards and is hinged at the bottom so as to cover most of the ladder opening when it is swung down. It is 3 ft. 10 in. long and its top comes even with the center of the set above the sollar set. It is held closed against the divider

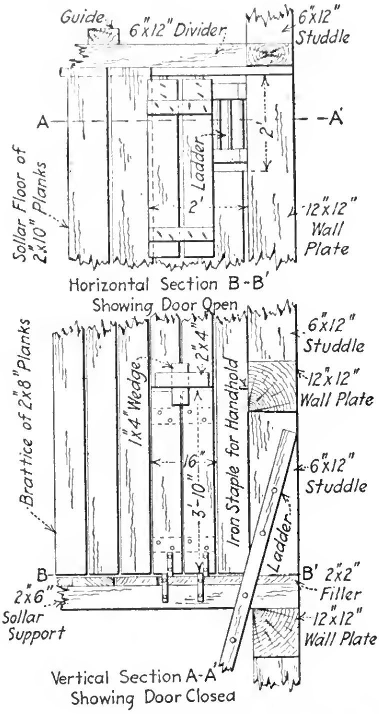


FIG. 82.—CORNER OF MANWAY COMPARTMENT.

of this set by a wedge made out of a 2 × 4-in. piece, slipping through a 1 × 4-in. slot in another 2 × 4-in. piece spiked horizontally to the brattice and divider. A 2 × 2-in. filler is nailed to the top of the 2 × 6-in. sollar joist next to the skipway, in order to bring the bottom of the door flush with the sollar floor. It is shown closed in the lower part of Fig. 82 and swung down in the upper part of the same illustration.

Inclined Shaft for Timber (By L. D. Davenport).—For a small mine on the Mesabi, where the overburden is shallow, the inclined timber shaft or timber slide seems to have several advantages over the vertical shaft generally used for this purpose. Only one top man is required to send

down timber instead of two; no rope, windlass, or headframe is needed; and an easy, safe entrance to the mine is had.

This description applies to the latest inclined timber shaft sunk by the Oliver Iron Mining Co., although there are a number of inclines in use at other mines in this district which are similar. Such a shaft has to serve two levels. Fig. 83 shows the arrangement of the sets where the incline intersects the top sublevel. A door made of double thickness of 2-in. plank deflects the descending timber or lagging to the top sublevel. When it is desired to send timber to the lower level this door is raised by means of a block and tackle fastened to one of the caps as shown. The free end of the rope is led to the stairway side of the shaft, where it is pulled. A vertical ladder against the side of the shaft connects the sub-

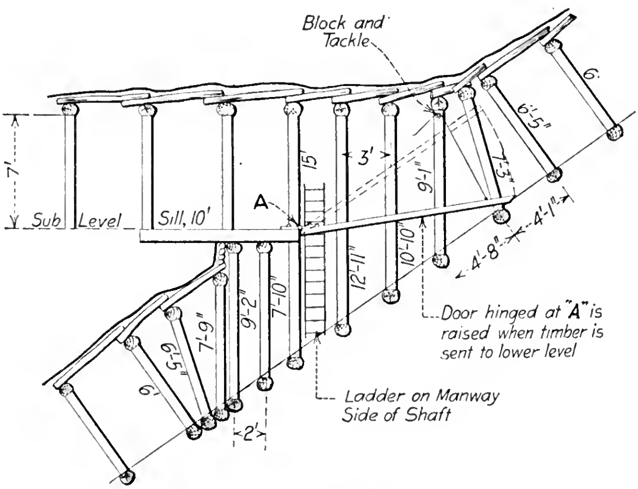


FIG. 83.—ARRANGEMENT OF TIMBERING AT SUBLEVEL.

level and the stairway. Fig. 84 shows the details of the shaft timbering, the stairway and the timber slide. This shaft has a dip of 35° and is 110 ft. deep along the slope. Timber travels down the slide faster than is necessary. In another mine the timber tends to hang up on a 90-ft. incline which dips at 29°.

Fig. 85 shows the details of the headframe used in sinking the shaft. Two 9 × 16-in. by 28-ft. timbers supported by two bents of round timber carried the rails. The dump was made of 6 × 8-in. timber faced with 5/8 × 3-in. flat iron. The skip was made from a tram-car box with the end door removed and bail and wheels fitted to it. The rear wheels had a wide tread to pass the dump and the sides of the box were cut back as shown.

The shaft was sunk through the overburden to the ore. To avoid

7

hoisting any ore a raise was holed through from below and trimmed out; the timbering was then carried to the bottom. When the shaft was completed the headframe was removed and a small shed with three walls and a roof was built over the collar.

Unwatering and Equipping Untimbered Shaft (By Douglas Muir).—Unwatering the Rayas mine of the Guanajuato Reduction & Mines Co., at Guanajuato, Mexico, was performed by bailing through the old Rayas main shaft. This shaft is circular in plan and has a minimum diameter of 36 ft., the usual diameter being 40 ft. A tunnel driven for drainage purposes cut the shaft 269 ft. below the surface and was chosen as the discharge point for the bailers. One steam and one electric hoist were used for the bailing, handling four bailers in all. At the time of beginning work, the water stood 770 ft. below the shaft collar, leaving 495 ft. to be unwatered.

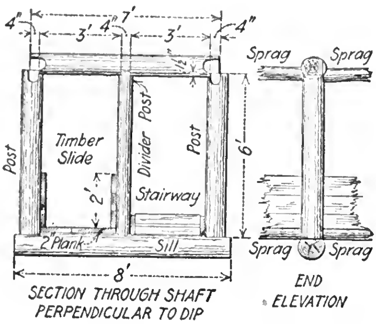


FIG. 84.—DETAILS OF SHAFT TIMBERING.

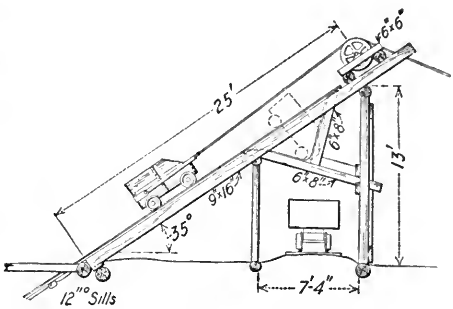


FIG. 85.—HEADFRAME USED IN SINKING.

Plumb lines were dropped from the sheaves of the two hoists to determine the exact plane of the wire-rope guides on which the bailers were to run. From the plane thus determined, the framework backstops *A*, Fig. 86, for receiving the kick-back from the bailers on discharging, and the apron decks for receiving and carrying off the water, were located, proper clearance for passing of the bailers being allowed. To support these structures, holes were drilled in the sides of the shaft and 1½-in. eye-bolts were wedge-driven and cemented into them. From these eye-bolts heavy wire cables were stretched across the shaft and tightened by turnbuckles, thus giving a support on which to lash planks to work from. The timber frames *A* of 8 × 8-in. material were erected back of and parallel to the plane of each set of guides to form the backstops. These frames spanned the distance between the two outside guides and were 16 ft. high, allowing for a considerable error in overwinding so that if such occurred, the bailer would not swing back toward the center of the shaft and strike the top of the backstop on its return trip downward.

The frames were suspended from heavy eye-bolts in the sides of the shaft 30 ft. above, thus giving a nearly vertical pull at the supporting points. On the faces of these frames were spiked blocks of wood and across these blocks were nailed 3-in. planks *B* to receive the blow of the discharging

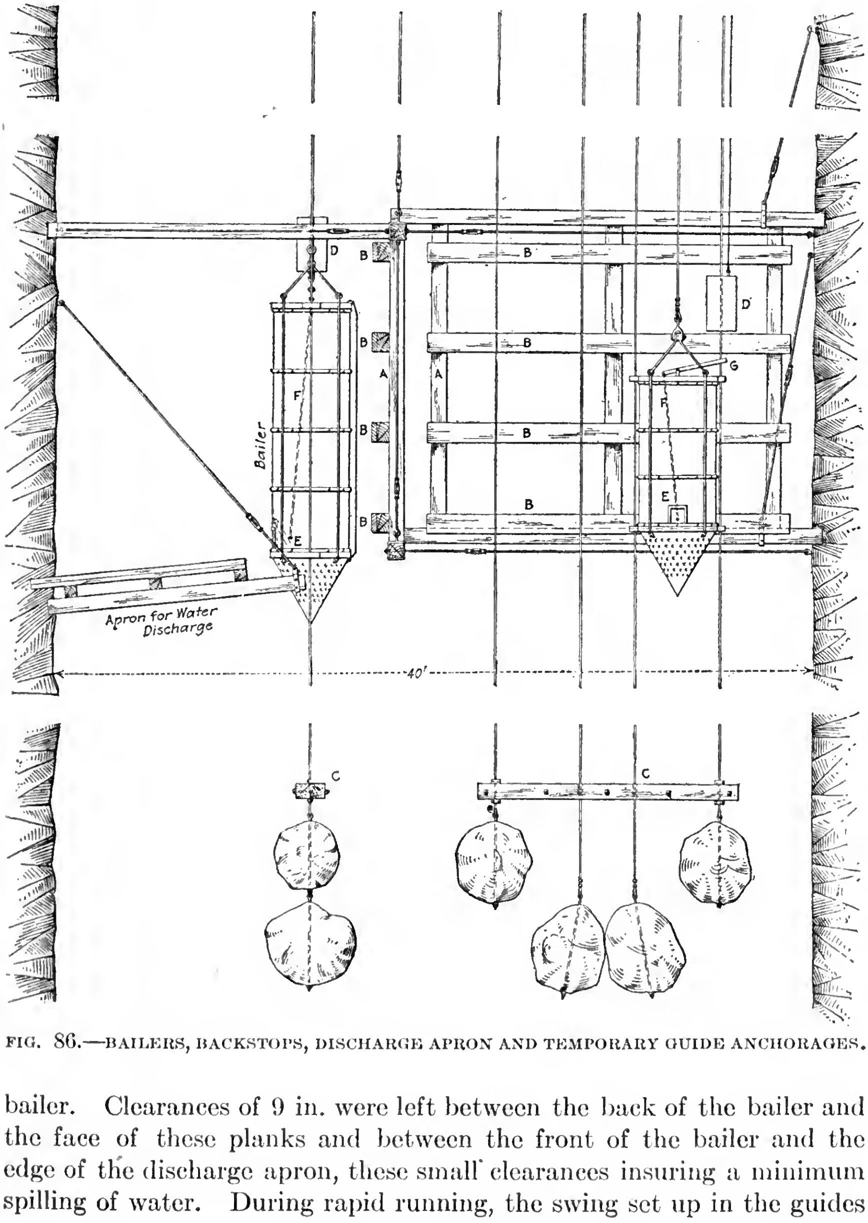


FIG. 86.—BAILERS, BACKSTOPS, DISCHARGE APRON AND TEMPORARY GUIDE ANCHORAGES.

bailer. Clearances of 9 in. were left between the back of the bailer and the face of these planks and between the front of the bailer and the edge of the discharge apron, these small clearances insuring a minimum spilling of water. During rapid running, the swing set up in the guides

often brought a bailer into violent contact with the planks. These were knocked off and no damage done to the bailer, which would not have been the case with a rigidly braced frame placed close to the bailer and without the rather loose planks.

The discharge decks were built of 2-in. planks on a 6 × 6-in. frame hung from the side of the shaft, spanning the distance between the two outside guides and having a steep slope to carry off the water quickly.

The guides are ⅞-in. galvanized wire rope. The drums on which they were carried were placed near the shaft at the surface, the ends of the ropes passed through clamps to hold them, and the ropes laid out until the ends reached the discharge station 269 ft. down. The guides for each bailer are spaced 4 ft. 6 in., with a 2-ft. 10-in. clearance between inside guides of the pair electrically operated and a 3-ft. 5-in. clearance for the steam-operated pair, since the latter were longer and heavier bailers and had more movement when in action. Rocks weighing about a ton each were drilled for eye-bolts, clamped one to each guide end at the discharge station and swung out into the shaft. Two 8 × 8-in. timbers were notched to match the spacing of each set of four guides and clamped to them by bolts passing through the timbers as shown at *C*, Fig. 86. The notches for the middle guides were made large, to allow them with their weights to be lowered one at a time by sliding freely through the timbers. The two outside guides with their weights were clamped tight to the timbers as a single unit.

The work of lowering the guides was carried out as follows: Each middle guide was clamped to a hoisting cable at a point below its sheave and lowered independently, sliding through the timbers kept hanging at the discharge station by the outside guides. When the end of the hoisting cable reached these timbers, the guide was made fast on the surface, the hoisting cable raised and the operation repeated until the two suspended rocks were about 115 ft. below the surface of the water. One hoisting cable was then unwound down the shaft and wound back on its hoist drum in the same direction as the other cable and the drums clutched in so as to have two cables to lower by at the same time. The hoisting cables were clamped simultaneously to the two outside guides, which were then lowered at the same time, in the same manner as described for the single guides. These guides were lowered until the clamping timbers were about 100 ft. below the surface of the water and 15 ft. above the rocks on the middle independent guides. When bailing had lowered the water-level nearly to the timbers, the guides were lowered another 100 ft. In lowering, the slack on the guides from their drums to the discharge station was handled and held by taking two wraps of guide cable around the hoist drum over the layers of hoisting cable already there.

The bailers on the steam hoist had a capacity of 1015 gal. net and the electric bailers 575 gal. They were built of No. 14 galvanized sheet iron, riveted in the form of a cylinder, having heads and bottoms of double 2-in. plank, fastened in by large wood screws. They were bound with bands of $3/8 \times 2$-in. strap iron for stiffening, and slung on the hoisting cables by means of a harness of iron rods and a bridle as shown in Fig. 86, representing a bailer of each size hanging at the discharge station. The bottom of the bailer had a 12×14-in. hole for an intake, over which was a hinged leather-covered door of mesquite. In the side at the bottom was a 10×10-in. hole to which was fitted a frame carrying a similar door E for discharge. The discharge door was connected by a 1/2-in. iron rod F to one end of a lever G mounted on top of the bailer. The other end of this lever was split and ran on the outside guide so as to engage the tripping weight D and effect the discharge. The tripping weight was a block of oak, slotted to fit on the guide, on which it slid loose, being hung from above by a small wire cable. In case the hoistman pulled by the discharge point, the weight was simply carried up along the guide and no damage done to the bailer. The guide blocks for the bailers were of cast zinc, in two halves, held in strap-iron frames bolted to 3-in. wooden extension pieces. On the bottoms of the bailers were cones of 1/8-in. sheet iron perforated with 1-in. holes, sufficient in number to give an area exceeding that of the intake door. These cones did away with all jar as the bailers struck the surface of the water.

A gin pole, mounted on the surface at the edge of the shaft, fitted with pulley and cable running to a geared winch, was used for removing the bailers from the shaft for repairs, spare bailers being kept in readiness for instant changing.

In July, an average month, 39,000,000 gal. of water was raised an average distance of 568 ft. at a cost of about 1/100 ct. per gallon. The mine was unwatered on Sept. 15, after removing about 150,000,000 gal. of water. Four and a half months of rapid, continuous bailing accomplished the task.

Upon completion of the unwatering, the work of equipping the shaft was begun at once. The rock weights and bottom timbers were removed and the guides anchored well below the bottom station as shown at A in Fig. 87. At the surface, eye-bolts of 1 1/2-in. iron, threaded for 3 ft. for tightening the guides, were hung through plates on the headframe, a structure built of steel shapes flat over the collar of the shaft, and the guides drawn tight with them.

An 11×11-ft. crosscut tunnel was driven from the grade of the mill electric railway, to pass by the shaft at a distance of 20 ft. At a point opposite the shaft, a raise 8×15 ft. in section was driven for 30 ft., forming an ore pocket for delivering ore to the cars, and from the top of

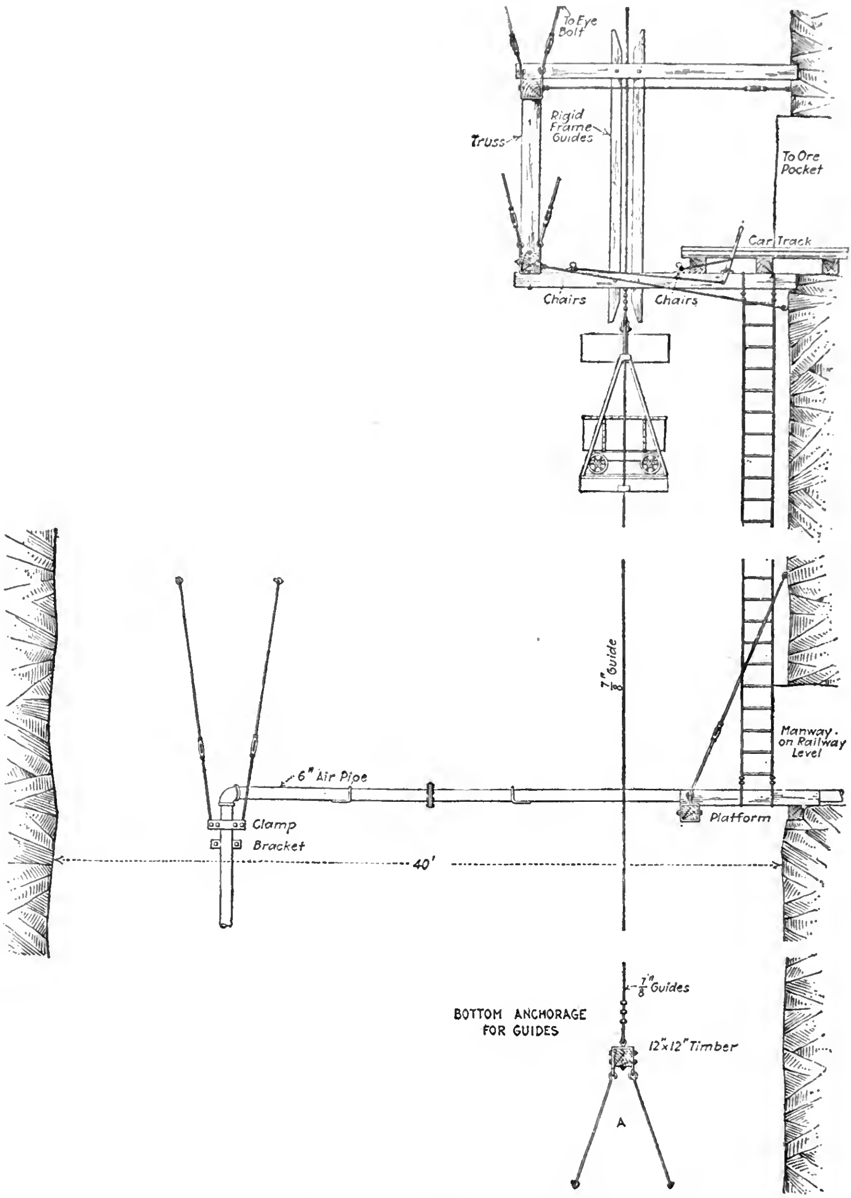


FIG. 87.—STATION FOR DISCHARGING ORE CARS, MANWAY STATION AND PERMANENT ANCHORAGE FOR ROPE GUIDES.

the raise a short crosscut 7 × 15 ft. in section was driven to cut the shaft. From the main crosscut, another small road was driven to the shaft for handling men and supplies. On breaking through, a timber platform was hung in the shaft in front of each connection point in order to catch the rock which otherwise would have fallen down the shaft and wrecked the discharge station. Both entrances to the shaft were arched with cut stone laid in cement mortar. The shaft was equipped for hoisting ore with the electric hoist only.

In front of the crosscut to the top of the ore pocket a truss of 10 × 10-in. timber was hung, back of the guides and parallel to their plane, by means of 1¼-in. rods to the end points of the top and bottom chords. These rods had turnbuckles, and were fastened by plate clevises to 1½-in. eye-bolts placed in the sides of the shaft 9 ft. above the top chord of the truss. To the bottom chord 10 × 10-in. timbers were bolted and seated in hitches in the side of the shaft. On this structure were placed the rigid-frame guides into which the cages run at landing, the chairs for landing, and the decking for the cars. In front of the manway opening, a platform was hung in the shaft extending to meet the edge of the cage.

Owing to the fact that the stations below were situated on the opposite sides of the shaft, it was necessary that at one of them the mine cars cross the shaft to reach the cage. Two trusses were hung in the shaft to form a bridge for tramming across to the cage, the cars entering the cage from one side and leaving it from the other. The trusses were hung similarly to the one described.

CONCRETE SHAFT LINING

Concrete Lining of the Kingdon Shaft (By Charles B. Eades and F. E. Calkins).—The reinforced-concrete lining of the two-compartment Kingdon shaft of the Old Dominion Co., at Globe, Ariz., completed on Aug. 28, 1912, extends from the collar to the bottom of the shaft, a distance of 1017 ft. In October, 1911, the timbering of this shaft was destroyed by fire, and it was decided to put in a concrete lining by lifts or sections, beginning near the top and working downward; the work was finally accomplished in six sections, from 150 to 220 ft. in height, depending on the condition of the ground. The work was done by contract.

About a month was spent in the preliminary work of erecting a temporary headframe, crushing plant, concrete mixer, etc., and in cleaning down the charred timbers and loose rock from the first section, extending from the collar to a point 160 ft. below. Two specially built, heavy, wooden cages swinging freely in the shaft were used throughout the work. In lining a section, the walls were stripped of charred timber and loose rock, beginning at the top and working downward, and light

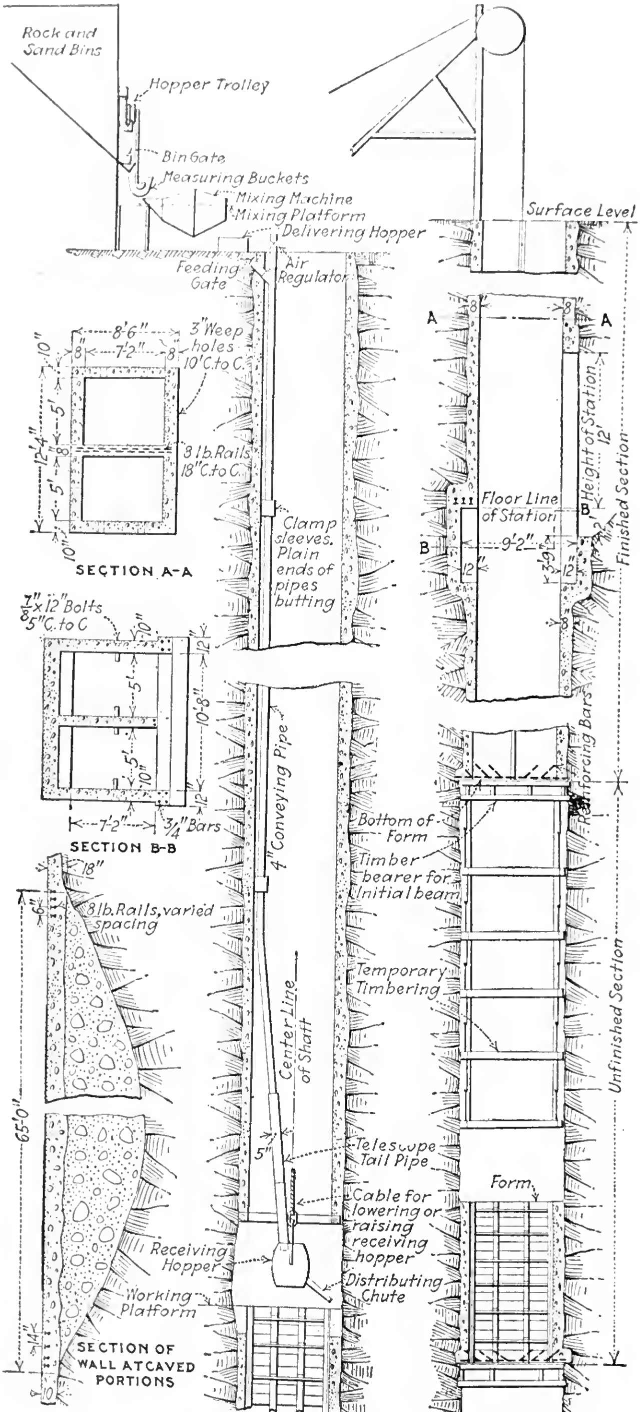


FIG. 88.—CONCRETING THE KINGDON SHAFT AT GLOBE, ARIZ.

temporary sets of timber put in, with a few lagging pieces wherever necessary, so that the men were always protected from falling ground. The general layout and design of the lining is shown in Fig. 88.

When the bottom of the section was reached, temporary timber bearers were placed along the sides and ends and across the center of the shaft, forms were built upon them, and a permanent reinforced-concrete bearer, 4 or 5 ft. high, was put in. Two or three days were allowed for this to set, after which the concrete lining was built up on top of it. The forms were built in sections 12 ft. high, and the concrete poured in between the form and the rock walls of the shaft. As soon as one 12-ft. section was filled, another was erected on top of it, plumbed and blocked, and filled with concrete in the same manner. The work proceeded thus until the bottom of the finished lining above was reached, the temporary timbering being removed as fast as the forms were erected.

The concrete was made to run from the conical mixer at the surface into a hopper and down the shaft through a 4-in. iron pipe to the point where it was needed; there it was caught in an ordinary steel sinking bucket suspended from the finished portion of the lining above, and allowed to run through a hole cut in the side of the bucket a few inches above the bottom, and through a short steel chute into the forms. This was an efficient and flexible arrangement, as the bucket could be easily swung or turned and a continuous stream of concrete directed to any part of the forms desired. The concrete was successfully dropped in this manner for a distance of over 1000 ft. in building the last section. Its falling velocity was but little greater through the pipe to the deepest part of the shaft than to the sections near the surface, provided the mixture was fed regularly and the upper end of the pipe closed at the end of each batch. This was easily done by placing a piece of canvas over the opening, thus stopping the air current in the pipe, checking the velocity of the concrete, and tending to break it up into smaller pieces, which fell with less force.

The forms used were of 2-in. lumber dressed to uniform thickness. They were set up by placing 4 × 4-in. posts in the corners of each compartment, with one 4 × 6-in. post between corners on the 5-ft. side and two on the 7-ft. side. A set of 2 × 6-in. braces was placed between these posts at every 3 ft. of elevation. In placing the forms two sets of false timbering were removed, giving room for a set of form posts 12 ft. long. These were put in, plumbed and braced and the 2-in. plank nailed to them a foot or so in advance of the concrete as it was filled in. When the form was full, two more sets of timbering were removed and another set of form posts put in place, so repeating the operation until the entire section was completed. Forms were left in place in each section until the entire

section was completed, then taken out, beginning at the top, and cleaned for use in the next section.

The specifications called for forms behind the concrete and back filling behind these forms for all cavities of too great size to be filled with concrete, but as the work proceeded the contractors realized it would be cheaper and more efficient to build rubble walls instead of back forms, filling in all back space with broken rock and boulders of sizes easily handled. When this rubble work was used the thickness of concrete walls was increased to as much as 18 in. in some instances and reinforced, the thickness of the wall and the quantity of reinforcement depending upon the vertical height of back filling, which in one instance was about 65 ft.

The mixture to be poured between the rubble walls and the form was mixed dry enough that no water would appear upon the surface after being well worked in a plastic mass that would run slowly down a slope of 20°. This dry mixture went through the pipe as readily as if wetter. Had the thinner mixture been used the water would have separated from the sand and rock, taking the cement with it and leaving a lean concrete behind.

The long walls of the lining were given a minimum thickness of 10 in. and the short walls and center partition, 8 in. Where the rock walls were very irregular, forming large cavities, large rocks brought down on the cages were thrown into the concrete. The center partition was reinforced every 18 in. vertically with mine rails laid across the shaft in the center of the wall. The end and side walls were thus reinforced only at points where the ground was bad and at stations. Bolts were imbedded in the concrete at regular intervals for fastening the guides. Old water pipes were cut into short lengths and placed in the concrete for weep holes through the walls. A set of four of these was placed every 5 ft. The best time for placing these pipes was at the end of a run of concrete, when they could be placed and pushed down until covered with concrete, without being filled.

The concrete was a 1:3:6 mixture of portland cement, quartz sand, and crushed limestone from ¼-in. to 1-in. mesh. About 2300 cu. yd. of concrete and 750 cu. yd. of large rock were used. About 25 men were employed on the job, working two shifts. The total time consumed was eight months, of which the first month was spent in rigging up the surface plant, etc., and most of the rest of the time in cleaning down the walls and putting in temporary timbering preparatory to concreting. In addition, the lower 110 ft. of the shaft consisting of a raise driven to one-compartment size had to be enlarged to full size. The actual work of concreting was done in 100 days, or about 40 per cent. of the total time consumed. Ordinarily, one 12-ft. section of concrete per day was put

in, but in bad ground a 6-ft. form would be used. The maximum day's work was 22 ft. The two compartments are 5 ft. by 7 ft. 2 in. inside the finishing lining. The shaft is used only for ventilation and for hoisting and lowering men working in the east end of the mine.

It is most interesting to note that the area of the compartments is over 60 per cent. greater than the area of the old, timbered compartments, which were 4 ft. by 5 ft. 6 in. showing that with a given area of ground broken a much larger shaft area may be obtained with a concrete than with a timber lining. The contractors received $30,000 for their work, or $28 per foot.

Concreting the Junction Shaft (By Robert H. Dickson).—The Calumet & Arizona Mining Co., of Bisbee, Ariz., replaced the timbers of its Junction shaft with concrete. The new shaft, 27 ft. 3 in. long and 6 ft. wide, was concreted to the surface from a point 1535 ft. below in 8 months and 24 days. Except at the stations, standard sets of forms, which could easily be set up and taken apart, were used. The materials used for the concrete were stored in overhead bins, from which they could easily be fed to the concrete mixer, Fig. 89. The 1:3:5 mixture was dropped through an iron pipe to that part of the shaft where it was to be placed. Steel reinforcement of 3/4-in. rods with a tensile strength of 50,000 lb. per square inch, was used in the curtain walls which separated the shaft compartments; in the columns required at stations; and in the shaft walls, where back-filling was necessary.

The shaft as concreted is 18 in. wider and 13 in. longer than the old shaft, the dimensions of all the compartments being increased. The relative size of the old and new shafts is shown in Fig. 90, compartments 1 and 5 being pipe compartments; 3 and 4, the main hoisting compartments; and 2, the dinky cage compartment. Concrete curtains separate the compartments, except that between Nos. 1 and 2 wood is used.

In the walls, the concrete was placed usually to fill all the space between the forms and the solid rock. Where the rock opened out so as to make necessary a wall over 2 ft. thick, a dry wall was built of rock and loose rock was thrown in behind. The concrete wall was never less than 8 in. thick, except where a rock of less than 4 sq. ft. in area projected no closer to the form than 4 in. At 10-ft. intervals, 2 × 5-ft. windows or air vents were left in the curtain walls to obviate the suction which would be created by the cage moving in so long a tube. Scaffolding boxes were left every 5 ft. in the pipe compartments, so as to permit staging to be erected at any part of the shaft. I-beams for carrying the pipes were set every 20 ft., and others for electric cables every 40 ft. from the 1510 level to the 770, and every 100 ft. from the 770 to the surface. The reinforcement in the curtains was placed horizontally about every 2½ ft., one rod being close to one side of the curtain, the next to the other.

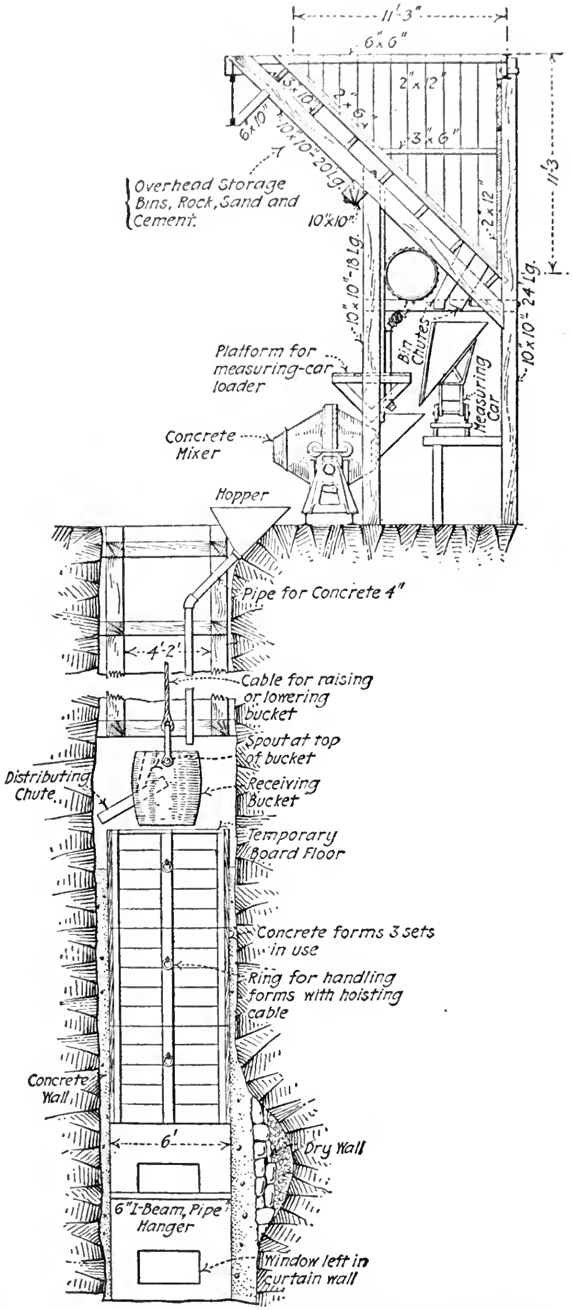


FIG. 89.—VERTICAL SECTION, JUNCTION SHAFT, SHOWING MANNER OF OPERATION.

The ends of the rods were bent at 45° to give a grip on the side walls.

The forms consisted of sections 5 ft. high. Three complete sets of

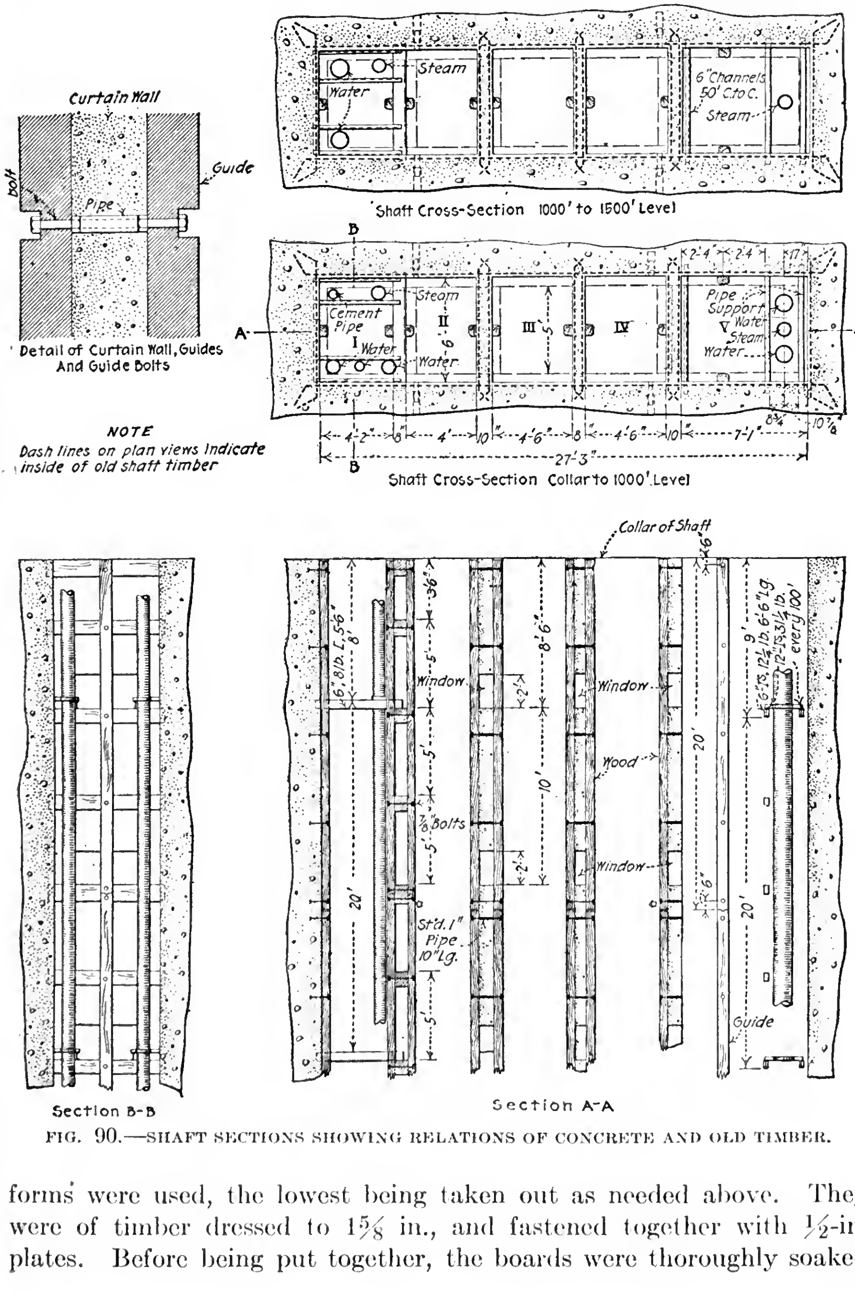


FIG. 90.—SHAFT SECTIONS SHOWING RELATIONS OF CONCRETE AND OLD TIMBER.

forms were used, the lowest being taken out as needed above. They were of timber dressed to 1⅝ in., and fastened together with ½-in. plates. Before being put together, the boards were thoroughly soaked

in water so as to be in the same condition while making, as they would be while in use. The forms consist of four pieces for each compartment, or more, in the case of those compartments where pipes in the shaft did not allow the handling of large pieces. In general braces of 2 × 3 × ¼-in. angles, were used across the compartments between the sections of the forms, to stay them against bending and movement.

In No. 4 compartment, typical of the others, the forms consisted of two sides, marked *A* in Fig. 91 and the two ends, *B*. The boards of each piece were fastened together with 4-in. channel at the center and a ½-in. plate at each end. Horizontal angles at the top reinforced the forms against buckling and also supported the boards of the working floors. As the angles were below the top of the form, these boards did not project over the curtain walls and were held in place. All four corners of the

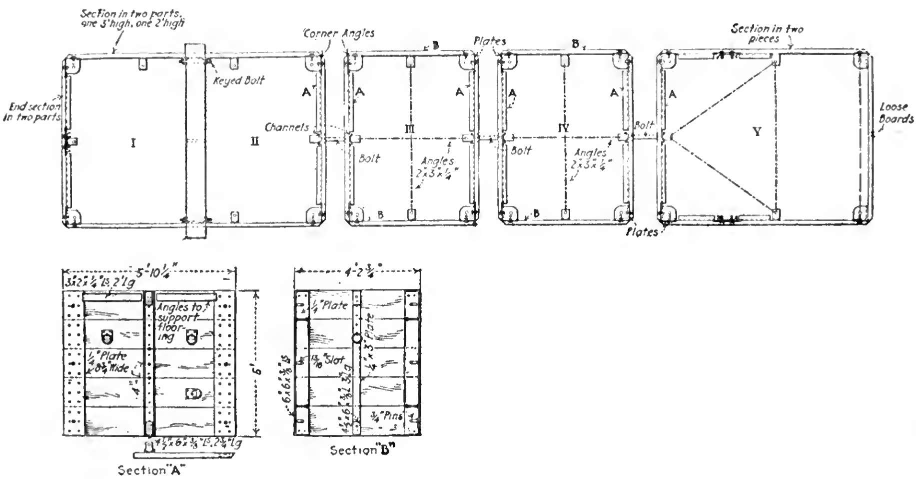


FIG. 91.—ARRANGEMENT OF THE FORMS IN THE SHAFT AND DETAILS OF TWO SECTIONS.

set were beveled to allow easy removal from the concrete. Each piece was provided with a ring for easier handling. The four pieces were held together by a 6-in. angle iron at each corner of the compartment. In compartment No. 1, the end was made in two pieces so as to allow it to be slipped out from behind the pipes. Compartments No. 1 and 2 are really one large compartment, divided by a timber partition. Timber was used instead of concrete in order that the two compartments might be thrown into one, if desired later. In several cases, as in No. 1 and 2 compartments, a long side consisted of two pieces fastened together by bolts and keys, as shown. The end of No. 5 compartment consisted of loose 12-in. boards because the pipes at that end of the shaft would not permit any other arrangement. A bolt through the curtain wall held the forms of each compartment to those of the adjoining.

Concrete columns 16 in. square took the place of walls at the stations where openings had to be left in the sides of the shaft. Curtain walls extended between the two opposite columns, as in the case of the walls above. The columns were reinforced with ¾-in. steel rods tied with wire. At the 1400 level an ore pocket 17 × 17 × 28 ft. was concreted. Forms were made underground as needed for the station columns and this pocket, and the concrete was poured into these in the same manner as with other parts of the shaft.

A 4-in. conveying pipe extended the whole length of the shaft and terminated 4 in. above the top of the receiving bucket, an ordinary sinking bucket with a spout attached. Concrete from the mixer was dropped through this pipe into the bucket, and thence poured through chutes so as to fill all the space between the forms and the rock walls. The cycle of operations involved concreting, removing timber, pulling forms from below and setting them up above. After a set was filled with concrete, a board floor was placed across the top of the last set of forms and upon this the men worked in removing the shaft timber.

Only enough timber was taken out at a time to permit concreting. One man split the nuts of the 1½-in. hanging bolts of the set above with a compressed-air cutter, while the concrete was being poured. At first, several of the dividers were chopped in two and pulled out of place by a chain fastened below the hoisting cage. This loosened the wall plates, which were in two pieces; thus the posts could be removed. Then by fastening the chain to the wall plates, they were pulled from their position. As each piece of timber was dislodged, it was placed on the cage. The removal of the timbers often dislodged some loose rock. In the case of bad ground, before a portion of the set was removed or immediately after, long pieces of lagging or stringers were placed up against the walls so that their upper ends could be blocked behind the last set of timbers in place, if possible, and their lower ends held in place against the wall by stulls or braces extending across the shaft. Often the stringer or lagging was held in place by several stulls. Except at stations, the open ground above a set to be concreted never exceeded 10 ft. As concreting proceeded, these stringers or lagging pieces were removed, if possible.

After removing the timbers, the temporary floor was taken up and the forms were pulled from the lowest set in place, set up, and blocked. Two bulkheads were always kept below the men, one 5 ft. below the lowest set of forms and one 30 ft. below this. In pulling forms, two men loosened them below while two above set them in place. The only tools required were a wrench and short bar. Beside the three sets of forms, used in the shaft, two other sets were kept to supply repair parts. As stated, the four pieces of form for each compartment were held together by bolting to angle irons at the four corners of the compartment. In the end pieces,

the nuts were contained in the form, Fig. 92, so that by unscrewing the bolt, leaving the nut in the form, the corner angles could be speedily removed from the keyed bolt of the side pieces. The ends of the angles had one flange turned over and welded. After taking out the corner angles from below, they were taken to a point above the last set of forms in place and were fastened to the corner angles of this set by means of keys and bolts through holes in the turned-over ends. Then the sides were pried off from the walls below and fastened to a chain suspended from the hoisting cage, lifted, dropped into place and fastened again to the respective corner angles. The end pieces were similarly handled.

After setting up, the forms were leveled and blocked in place. A line was stretched across the shaft between two plumb lines, and the upper edge of one side set of forms was blocked out from the rock wall so that it came a certain distance from this line, and similarly with one end. The lower edge of the set was held in position by fastening to the set below.

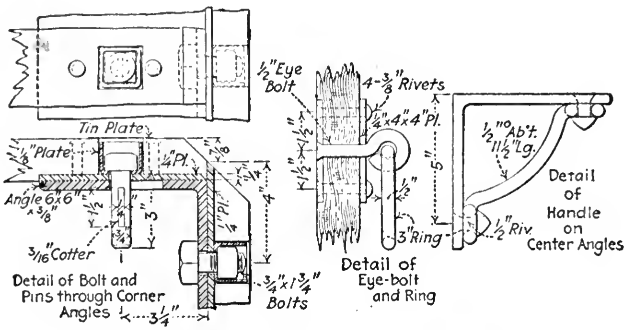


FIG. 92.—DETAILS OF SOME OF THE FORM FITTINGS.

The forms were carefully leveled by means of a straight-edge. Small iron wedges were placed under the corners of the forms to level them up, if necessary. Window and staging boxes were placed in the forms as required.

The bucket was usually hung in No. 2 compartment, the bottom 3 to 4 ft. above the top of the set. From the spout the concrete was conveyed to the point of deposit through chutes which telescoped like coal chutes used in cities. The bottom of the last set of timbers was usually 2 or 3 ft. above the top of the form. In dropping, the concrete was churned in the bucket and mixed more thoroughly than would be possible in the mixer. The concrete was tamped into place behind the forms.

Three shifts, each of eight men and a shift boss, made up the shaft crew. Each shift completed as much of a cycle as it could and the next shift started where the preceding shift left off. The time required to remove timber, pull forms and concrete varied exceedingly under the

changing conditions, such as the experience of the men, character of the ground, etc When first started, the work required as much as two shifts to pull forms, while later on the average time was 5 hr. During the months from February to June, it required an average of about 20 hr. to complete the cycle, the time being distributed approximately as follows: Removing shaft timbers, 5 hr.; removing waste or back filling, 4 hr.; pulling forms, lining and blocking, 8 hr.; concreting, 3 hr. For the upper 450 ft., the average time required for a cycle was 12½ hr. with an approximate distribution of time as follows: Removing timber, 2 hr.; drilling and shoveling, 4 hr.; pulling forms, lining and blocking 5 hr.; concreting, 1½ hr. For this instance, the walls had to be blasted all around an average of 8 in., making about 25 cars per 5 ft. of shaft. The actual concreting was finished Aug. 24, 1913.

Cost Sheet per Foot and per Cubic Yard from Dec., 1912, to Aug. 24, 1913

	1912, Dec.	1913, Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.
Yardage	444.8	900.3	933.3	1096.5	1120.2	1093.8	931.5	1000.2	853.5
Footage	60	140	170	185	180	200	170	235	196
Alteration in piping, per ft.	$7.84	$4.42	$4.09	$3.44	$5.45	$3.87	$4.42	$1.94	$2.35
Alteration in transmission lines, per ft.	0.74	1.37	0.58	1.14	2.36	1.16	0.54	0.57	0.98
Removing shaft timber, per ft.	19.40	9.52	6.66	7.66	9.27	8.77	7.08	4.27	3.87
Waste and filling, per ft.	4.37	3.88	2.57	4.09	4.41	4.68	10.75	7.11	12.06
Guides, per ft.	0.14	0.32	2.35	1.49	0.00	2.74	5.33	6.57	6.07
Hoisting, per ft.	29.26	12.15	10.42	10.37	9.96	9.78	a20.37	6.25	7.36
(1) Repairs to concrete pipe, per ft.	......	......	......	0.07	0.10	0.15½	0.12½	0.01	0.00
(1) Repairs to concrete pipe, per yd.	......	......	......	0.01	0.017	0.03	0.03	0.00	0.00
(2) Forms, per ft.	3.24	6.29	4.72	4.05	2.93½	2.15	2.25	1.60	1.85
(2) Forms, per yd.	0.44	0.98	0.86	0.68	0.38	0.39	0.30	0.38	0.43
(3) Cost of concrete materials, per ft.	53.83	43.52	26.00	30.31	21.33	23.68	24.93	18.50	19.66
(3) Cost of concrete materials, per yd.	7.26	6.77	4.73	5.11	3.43	4.33	4.55	4.35	4.53
(4) Moving forms, per ft.	27.17	16.37	11.63	10.18	8.93	8.57	8.20	7.11	7.00
(4) Moving forms, per yd.	3.73	2.55	2.12	1.72	1.43½	1.57	1.50	1.67	1.65
(5) Distributing concrete, per ft.	9.20	6.64	4.06	3.17	2.77	2.25	2.73	1.90	2.50
(5) Distributing concrete, per yd.	1.24	0.92	0.66½	0.53	0.44½	0.41	0.50	0.45	0.58
Concrete totals 1-2-3-4, per ft.	93.44	72.82	46.41	47.78	36.07	36.80	38.23	29.12	31.01
Concrete totals 1-2-3-4, per yd.	12.67	11.22	8.38	8.05	5.72	6.73	6.88	6.85	7.19
Supervision, per ft.	b15.65	b6.81	1.49	1.39	1.39	1.30	1.50	1.00	1.00
Supervision, per yd.	2.11	1.06	0.27	0.24	0.24	0.14	0.27	0.24	0.23
Miscellaneous, per ft.	2.08	0.67	0.47	1.45	0.52	0.84	1.69	0.73	0.62
Total, per ft.	172.92	110.96	75.04	78.81	69.43	69.94	89.92	57.56	65.32

Total yardage, 8374.1; total footage, 1536.

a Includes cost of new hoist rope.

b The shift bosses during Dec. and Jan. were carried under "supervision."

The guides were bolted to the curtain walls. A short length of pipe was set in the curtain wall while concreting, with wooden blocks at its ends. When setting the guides, the blocks were removed, the guide bolt inserted and held tight to the pipe by nuts which were screwed on to occupy the cavities left by the blocks. The guides were placed over the bolts and held by nuts in recesses. The guides were set from a "go-devil," a box as long as a guide, of a cross-section to fit the compartment, the four corners formed of four long stringers held together by crosspieces and tie rods. On the crosspieces were laid floors, so as to form three decks on which the men worked. In this way three men at a time could bolt on a guide, one on each deck.

In the cost table, waste and filling include the labor and supplies used in enlarging the shaft, the removing of the waste dislodged while tearing out the old timbering and the labor and supplies used in refilling caves in the shaft caused by removing timber. Concrete cost includes the cost of quarrying and crushing the rock, the freight, sand, cement and supplies, the mixing and storing of the concrete, and the cost of reinforcement.

Rectangular Concrete Shaft Lining (*Coal Age*).—A shallow concrete-lined shaft sunk by the Bunsen Coal Co., of Danville, Ill., is illustrated in Figs. 93 and 94.

The thickness of the walls for 89 ft. down from the collar is 18 in., and thence to the bottom at 225 ft., 12 in. They are of plain concrete except the corners, which are reinforced vertically, each with two lines of $\frac{7}{8}$-in. by 16-ft. twisted steel bars, back of which are placed $\frac{7}{8}$-in. by 6-ft. bars laid horizontally and spaced every 2 ft. for the entire depth of the shaft. The dividers are all 10-in. I-beams weighing 35 lb. per foot.

The forms were composed of $1\frac{1}{2} \times 10$-in. tongue-and-groove yellow-pine boards, surfaced on both sides, and cut into lengths making the section $2\frac{1}{2}$ ft. high. Nailing strips $1\frac{1}{2} \times 12$ in. extended around the top and $1\frac{1}{2} \times 4$-in. pieces at the bottom formed cleats for the several units and held the matched boards securely in place. The top piece extended a distance of 1 in. above the top of the form on the inside for the reception of the next form section. For stiffening the forms and supporting the concrete until it should set, a steel frame consisting of $4 \times 4 \times \frac{1}{2}$-in. angles was placed midway of each section and attached to the form boards by means of $\frac{3}{4}$-in. bolts spaced 13 in. center to center around the entire section. The steel framework was stiffened at the corners with $4 \times 4 \times \frac{1}{2}$-in. angle bracing; while $\frac{3}{8}$-in. splice and gusset plates were used for bolting together the several units composing the frame. To insure easy removal of the forms and to increase the thickness of the walls, the corner pieces were cut on a 45° angle with a 5-in. bevel face. The vertical edges of each unit, including the forms and

angle frames, were also cut on a 45° bevel, while 6 × 10-in. pockets were made in the top of every other form for the reception of the 10-in. I-beam dividers which are spaced vertically 5 ft. center to center. The completed 2½-ft. section, including the steel frame, weighed 2080 lb., and the heaviest unit 400 lb.

No concrete was poured until the bottom of the shaft was reached. A temporary lining was used in sinking. The 8 × 10-in. dividers supporting the temporary lining were removed as the concrete lining

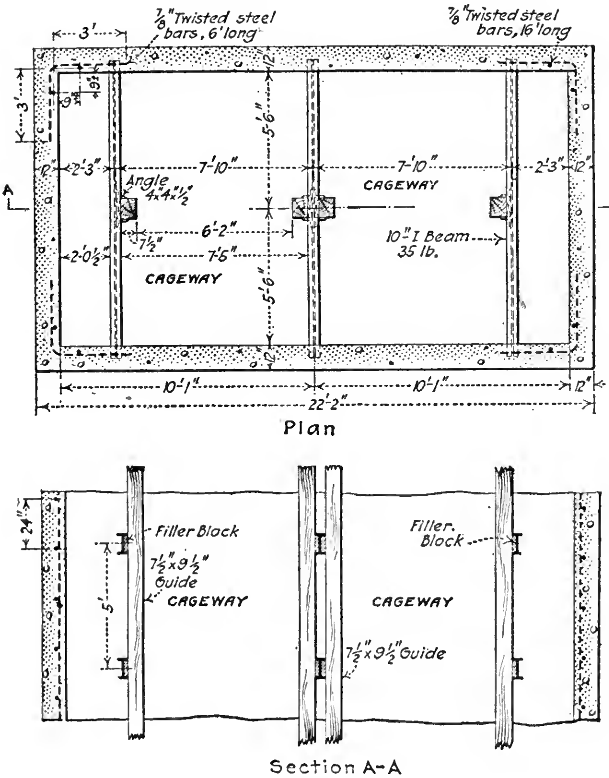


FIG. 93.—PLAN AND SECTION OF CONCRETE LINING.

advanced; these were replaced after each form was in position with 4 × 6-in. temporary supports for the forms until the concrete had set. Both lines of 4 × 6-in. temporary support were carried on 1½ × 4 × 10-in. blocks attached to 1½ × 10-in. vertical wall plates cut in 5-ft. lengths. These plates extended back of the forms. Four 1½ × 2½ × 10-in. blocks attached to the back of the wall plates cleared the steel angles on the forms and made a substantial support.

The concrete yardage per vertical foot for 12-in. lining walls was

2.5 cu. yd., and 3.8 cu. yd. for walls having a thickness of **18 in.** All of the concrete was composed of one part cement and five parts of clean river gravel. Concreting started Aug. 7, 1914, and the collar was reached Sept. 8. The entire work done in connection with the concreting of the shaft-lining walls occupied four and one-half weeks, giving an average of 50 ft. per week.

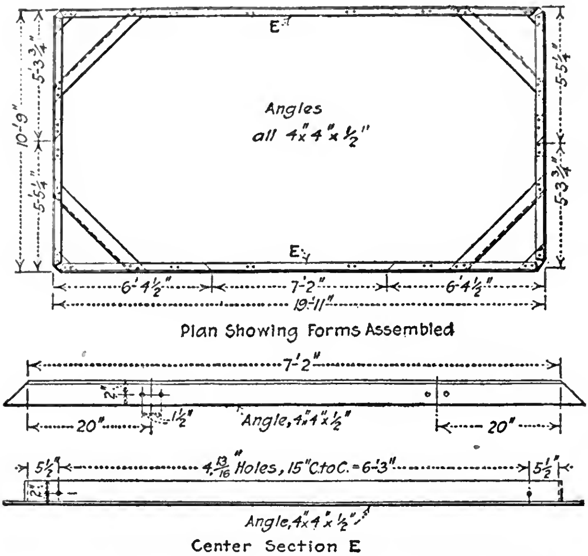


FIG. 94.—STEEL FRAME USED FOR STIFFENING FORMS.

Raising and Enlarging Negaunee No. 3 Shaft (Lake Superior Mining Institute).—The No. 3 shaft at the Negaunee mine, Negaunee, Mich., was started by sinking a test pit to solid rock through 68 ft. of sand and hard pan. As no water or quicksand was encountered, the main drift was driven from the bottom level of the old workings, 806 ft. below the surface, to the location of the new shaft. A raise was then commenced, to connect with the bottom of the test pit.

The general arrangements at the level are shown at 2, in Fig. 95. The cribbed compartment was divided in the center by 2-in. planks forming two compartments, one for pipes and ladder, and the other for a small bucket for tools and cribbing. In 3 is shown the method of supporting the cribbing, and the arrangement of the chutes. Below the dirt compartment a bench of solid rock, on the same angle as the lip of the chute, served as a permanent indestructible bottom. The cribbing was accurately framed from 6- to 8-in. round tamarack timber, each piece faced on one side to maintain dimensions inside the compartments. This

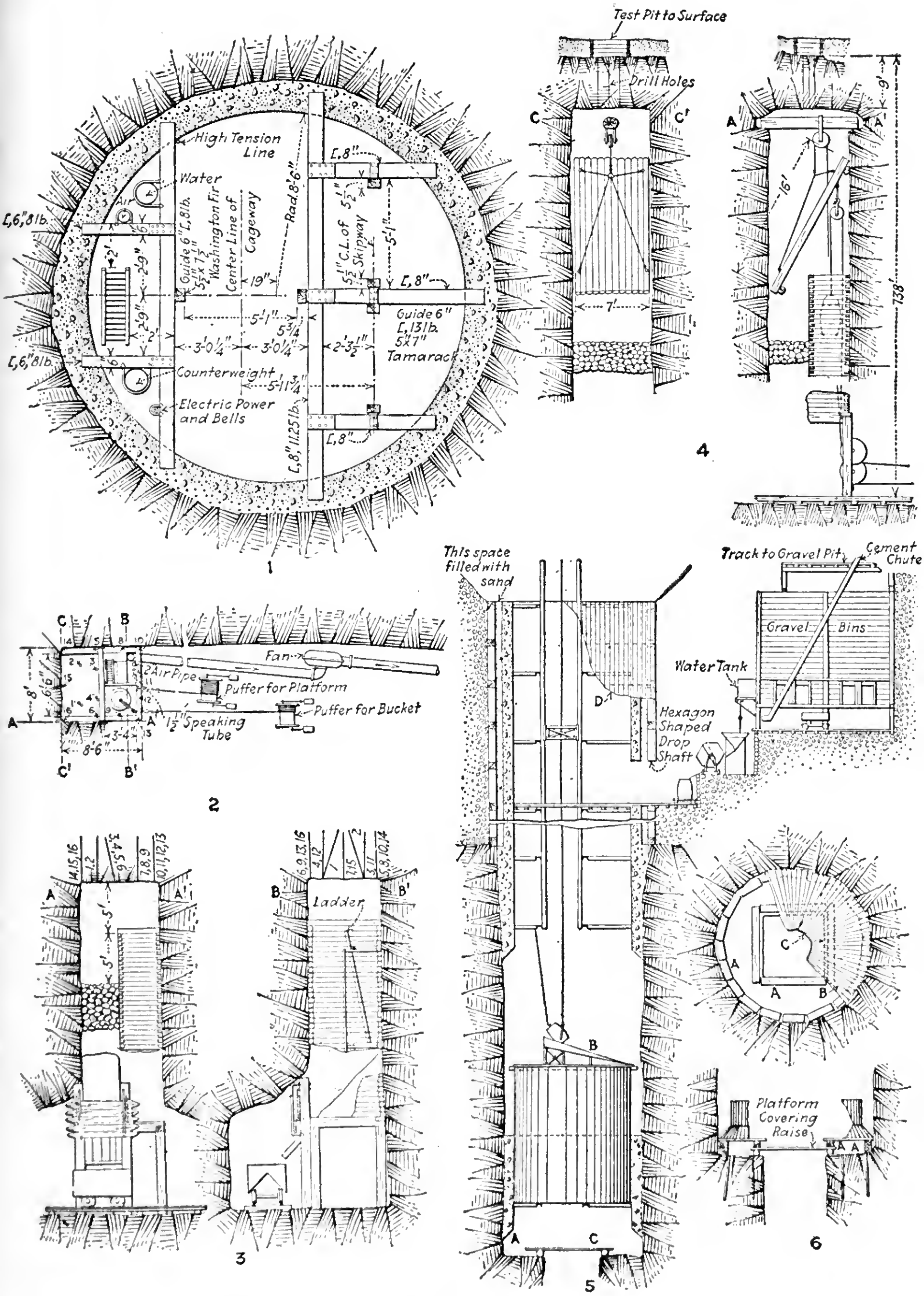


FIG. 95.—NEGAUNEE NO. 3, RAISE AND SHAFT.

greatly aided the miners in building up the cribbing plumb and made rapid hoisting practicable.

The crew was composed of nine miners, three on each 8-hr. shift. Stopers were used for drilling. It was not advantageous to blast a deeper cut than 5 ft. on account of the breaking or shifting of the cribbing. The best results were obtained by the round illustrated in 3. Before blasting, the cribbed compartments were covered with a 6-in. sollar, one small hole being left in the ladder-road side to allow a man to pass through after blasting. The round of 16 holes, was blasted with fuse in three sections as follows: (1) Holes 1 and 2; (2) 3 to 9; (3) 10 to 16. At intervals of about 200 ft., small drifts about 8 ft. long were driven from the ladder compartment, each drift protected by a door. A connection was made with the air pipe to prevent the possible danger of the men being knocked out if, for any reason, the fan failed to work. It would have consumed too much time for the men to go to the very bottom after the raise reached a great height. Before blasting the crew would enter these small drifts and suffered no discomfort even when the back was not over 30 ft. above them.

A small "puffer" was placed on the level and run by compressed air. After each cut was blasted and the loose rock in the back well trimmed, a stout pole was wedged across the raise as close to the back as possible. A 10-in. sheave was then hung from this pole so that the rope leading to the bucket was in the center of the hoisting compartment, the other end passing down one corner of the compartment to an angle sheave and thence to the puffer. Signals were given through a speaking tube, of 1½-in. pipe, extending from the puffer to the top set of cribbing. An oil barrel, reinforced in the bottom with wood and on the sides with steel strips, was used in preference to an iron bucket. On account of its depth tools and cribbing could be hoisted with little danger of their falling out. The bulge in the center lessened the possibility of the buckets catching in the cribbing.

On the level at a short distance from the raise a fan capable of exhausting 2040 cu. ft. per minute was set up; 10-in. spiral-riveted pipe with flanged joints every 10 ft. extended from the fan up into the ladder compartment. The upper end of the pipe was kept as close to the last set of cribbing as possible. The discharge pipe, 10 in. in diameter, of galvanized iron, was not perfectly tight and caused some trouble. It extended to an abandoned part of the mine, where the foul air was discharged behind a tight door, which prevented it from reëntering the main drift and contaminating the pure air needed in the raise. After blasting, the fan was started and the gases sucked out. It was found a great advantage to run a piece of old hose above the sollar and discharge compressed air directly after blasting, in order to hasten the removal of

the gases. The gases were all removed within 15 min., but as an extra precaution the men did not return to work for 30 min. It was not necessary to run the fan except while blasting.

On Jan. 1, 1909, the top of the raise was 20 ft. above the rail in the drift. On July 19, 1909, a hole was located in the back of the raise to strike the center of the test pit, the size of which was 2 × 3 ft. The first attempt was successful, showing that the raise had been brought up accurately. The total distance from the rail to the bottom of the test pit is 738 ft.

In order to avoid hanging up of the raise while stripping to full shaft section was going on, it was decided to remove all the timber before beginning stripping. For this purpose a platform, 7 × 16 ft., was made of 8-in. round tamarack bolted to crosspieces as shown in 4. A wire rope was attached to each corner so that the platform would hang at 65° from the horizontal. It was hoisted one piece at a time and bolted together at the top of the raise. A 10-in. sheave was hung from a strong timber at the back of the raise; the platform was slung from this by a 5/8-in. rope as shown, the other end of the rope passing through a slot in the platform and down the hoisting compartment to a small puffer on the level below. The platform served to protect the men from any possible fall of rock, the steep angle tending to prevent breakage. The cribbing timber as removed was lowered in a bucket supported by a sheave attached to the platform. The platform was always kept at the proper height above the workmen. Pipe, cribbing and plank were removed from the raise in 28 days.

A circular shaft was decided upon, 17 ft. in diameter and with a 1½-ft. wall, as shown in 1. The raise was blasted through into the test pit and sand was milled down to enlarge the surface excavation sufficiently to permit installing concreting equipment. The sand, however, packed in the raise bottom and as water accumulated came out in rushes. The rest of the sand was therefore hoisted by means of a bucket on a derrick. At 40 ft. below the original surface, a hexagonal wooden shaft, 24 ft. inside diameter, was built. The sets were 5 ft. apart with 3-in. planks spiked to the outside, *5D*. Alternate planks were cut 5 ft. 6 in. and 11 ft. in length, thus tying the sets together and developing a rigid structure. This shaft was allowed to drop slowly to the ledge as the material below was excavated.

A track was run at the surface to a nearby gravel bed and bins, water tank, mixer and a kibble on a truck installed as shown in 5. The kibble could be pushed to the center of the shaft and lowered on the hoisting rope to dump into the forms. A crew of five surface men handled 35 cu. yd. in an 8-hr. shift, sufficient for 10 ft. of shaft lining. The water was heated to boiling by exhaust steam and in winter the gravel was also

heated by steam pipes in the bin bottom. Cement was handled down a chute kept constantly full with the bags, the removal of one from below allowing the whole row to slide down.

The sides of the raise were stripped down with hand-hammer machines, the usual round being 18 holes, 16 ft. deep, blasted with delay-action electric fuses. As great a depth as possible was stripped at a time, the distance being a multiple of ten and varying from 20 to 90 ft.

The concreting forms were rings 5 ft. high and 17 ft. in diameter, in four sections. After a section was stripped, the engineer located the bottom of the lowest form. A base for the forms was made around the collar of the raise by means of a number of short pieces of timber properly leveled, 6*A*. A floor *B* was made of 2-in. hardwood planks, 7 ft. long, sawed to a radius of 10 ft., the radius of the rock section of the shaft. Small openings next the rock were stopped with small pieces of wood to hold the concrete. A 6-ft. hole remained in the center of the floor. One of the floor planks, *C*, was laid with the wide end in to make removal easy. The first form lowered was of special shape so as to leave the concrete bottom edge at an angle of 45°, 5*A*. A regular form was lowered and set upon this, being positioned by plumb lines from above. The steel dividers were lowered, positioned by plumbing and clamped to the forms. They thus served to support a working platform. The concrete distributing trough was next placed in position, extending from the shaft center to the edge of the forms, with a drop of 2 ft.

The kibble was lowered with ½ yd. of concrete and dumped into the trough by means of a hook on a rope, caught into a ring on the kibble bottom. Forms for another section, 10 ft. high, were next put into place together with another set of steel members. After completing the concreting of the 15 ft., requiring 48 hr., the floor under the lowest form could be removed, together with the lowest set of forms, for use higher up. The routine was for the morning shift to concrete 10 ft. of shaft one day and place 10 ft. of forms the next, the night and afternoon shifts stripping and preparing the next section. The size of the crew was figured so that both operations proceeded at about the same rate. For the section of shaft between the rock and the surface, special wooden forms were used as it would have required too much concrete to fill the space between the steel forms and the hexagonal shaft. These were rings of strap iron bent to a diameter of 20 ft. with 3-in. hardwood planks wired to the inside of them.

To cover the raise collar during stripping a 12 × 12-ft. platform, 5*C*, was used; this had wire ropes from the corners to a central ring and was hoisted and suspended from the lowest steel sets during blasting.

In making the joint between two sections of the concrete, the top of the form for the new section would be in line with the thin edge of the

45° bottom of the old section. A collar was then bolted to the form, its height 1½ ft., its upper edge turned toward the shaft to form a lip. Rich concrete was rammed in between the ring and the old face; the space was not filled in this first operation. After the concrete had set, the collar was taken off, all leaks between the old and the new faces caulked with oakum, and the small remaining triangular space filled with rich mortar. In this way all water was pretty thoroughly shut off.

Through the main stretch of the shaft, from the ninth level to the top of the rock, the work cost per foot for raising, $18.68; for stripping, $17.33; for steel dividers, $10.06; for steel forms, $1.26; for temporary surface structure and equipment, $6.74; for concrete, $17.22; for compressed air, $1.00; giving a total cost per foot, less salvage, of $72.17. The length of this stretch was 738 ft. and the speed of stripping and lining was 64 ft. per month. The total cost of the work between the rock and the surface was approximately $5890 and for the 158 ft. from the skip pit to the ninth level, involving winzes and special work, it was $17,840. The cost per yard of the concrete was $7.64; the average thickness of the walls was 16.2 in.

Steel and Concrete Lining of Palms Shaft (Lake Superior Mining Institute).—The recently sunk Palms shaft of the Newport Mining Co. is vertical, lined with steel and concrete. It is divided into five compartments, as shown in Fig. 96. The skip compartments are 4 ft. 10 in. by 6 ft., and the cage compartment is 6 ft. 2 in. by 10 ft. The outside dimensions of the shaft are 10 ft. 10 in. by 17 ft. 6 in. The 17-ft. 6-in. wall plates, the end plates and the two dividers, each 10 ft. long, are 5-in., 18.7-lb. H-section steel members. The other two dividers are 4 ft. 10 in. long and consist of 4-in., 13.6-lb. H-sections. The eight studdles are 3 × 3 × ¼-in. angle irons. Most of the sets are spaced 8 ft. center to center, but in heavy ground some sets are spaced 6 ft., and a few of them 4 ft. The wood guides are 5¾ in. by 7¾ in., and two of them, as shown, are strengthened by 7-in. channels. Hoisting was done with two 26-cu. ft. buckets weighing 900 lb., operated by an electric hoist. Another single-drum electric hoist handled a light cage for timbermen, running in the middle compartment of the shaft. The shaft passes through quartzite for part of its distance; the broken quartzite was saved and crushed for concreting purposes.

It was found possible when the rock would stand for 14 ft. below the lowest set, to rivet the steel members together on the surface, lower the set intact and swing it into place. Shoes on the two lower corners guided it to the shaft. Four 1-ton duplex chain blocks were used for swinging it into position. To each corner of the set was fastened a ½-in. sling chain about 3 ft. long, with a 5-in. ring on one end and a 3-in. ring on the other, to which the hooks of the chain blocks were attached. When a

14-ft. space could not be maintained, the sets were riveted in parts and bolted together below. When the rock walls were more than 8 or 9 in. from the sets, 4-in. tie timbers were placed vertically 4 in. outside the sets and about 2 ft. apart. Between the steel sets and these timbers, 4-in. wood blocks 12 in. long were placed. Outside the verticals, 1-in. rough boards were set horizontally to act as outside forms when it came time to pour the concrete lining. Between the boards and the solid rock, lagging was filled in. When the rock was less than 8 or 9 in. from the sets, 4-in. flat timbers were placed between the flanges of the wall plates and end plates, and lagging was placed behind these up to the rock. This lagging was left in place until concreting time, when it was removed from the shaft.

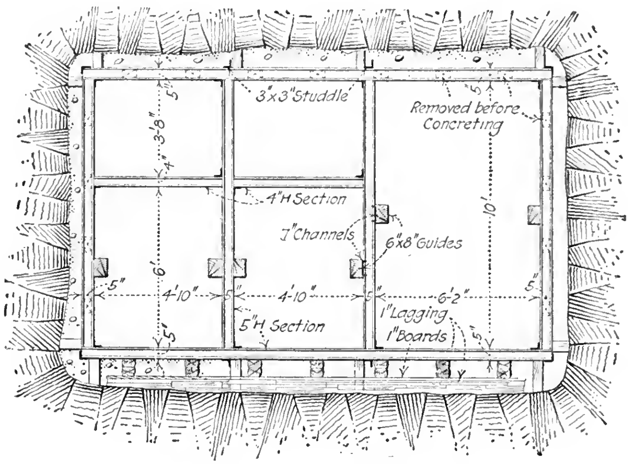


FIG. 96.—PLAN OF PALMS SHAFT.

During the process of sinking, at every 75- to 100-ft. point two or three adjacent sets were filled in solid to the rock with concrete, eliminating the necessity of bearers set in hitches. This concrete was mixed on the surface and lowered in a hopper, from the bottom of which a flexible spout extended. The hopper is shown in Fig. 97 and the spout in Fig. 98.

When a depth of 1207 ft. was attained, it was thought necessary to complete the concreting, because of the approach of cold weather. This concreting was started at a depth of 1170 ft. The concrete was mixed on the surface in the proportions of 1:3:5 and carried through a launder to a 4-in. flanged pipe down the shaft, which telescoped into a 5-in. branch, shown in Fig. 98. The branch took the blow of the concrete, and at its bottom it was connected with a reverse bend having its lower end vertical. An 18-ft. flexible spout fitted over this directed the concrete to the proper place behind the form. While the concreting force was filling one set,

other men were removing the blocking from the set above, hanging strands of old wire rope vertically 1 ft. apart and horizontally about 3 ft. apart, to be used for reinforcement, and placing the ouside forms. For an 8-ft. span, 2-in. hardwood planks were used, and for the 4-ft. and 6-ft. spans, 1⅝-in. hardwood planks. The planks were cut on a bevel at the upper end, so that the concrete came out underneath the steel set to act as a support. At the bottom end, in passing into the outside flange of the H-sections, 2-in. strips of wood about 12 in. long were laid 1 in. apart between the bottom end of the plank and the inside flange of the section. When these strips were removed, the planks were easily taken out. The corners were left open for concreting, so that a solid column of concrete was obtained in each corner over the entire depth of shaft. Furthermore, where lagging and timber were left between the

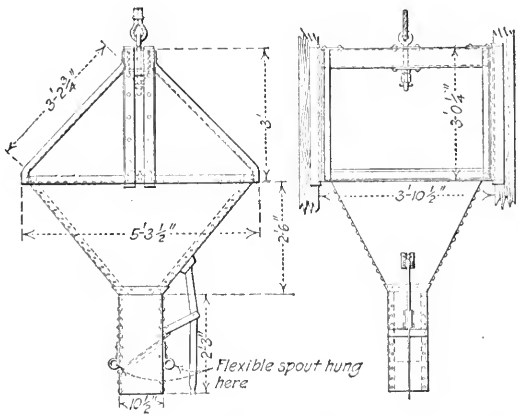


FIG. 97.—HOPPER FOR LOWERING SMALL LOTS OF CONCRETE.

concrete and the rock, openings for the concrete were left directly back of the wall plates and end plates to the solid rock, so that in all cases the concrete extended from the steel set to the rock. A $6 \times 8 \times 12$-in. block was laid in the concrete midway between the 8-ft. sets, to serve as a support for the two end guides.

Nine miners and one foreman per shift did the drilling, blasting and mucking and assisted the timbermen in placing the sets, concrete bearers and 12-in. ventilating pipes. Three timbermen per shift, with two foremen for the 24 hr., lagged the sets, put in the guides, extended the air lines, placed the ladders and substituted for absent miners. There were, furthermore, for each 24 hr., four engineers, two top-landers, two men to handle rock, etc., and two blacksmiths. The concreting required the 10 miners for removing lagging, placing reinforcement and placing plank forms. The four timbermen attended to the distribution of the concrete

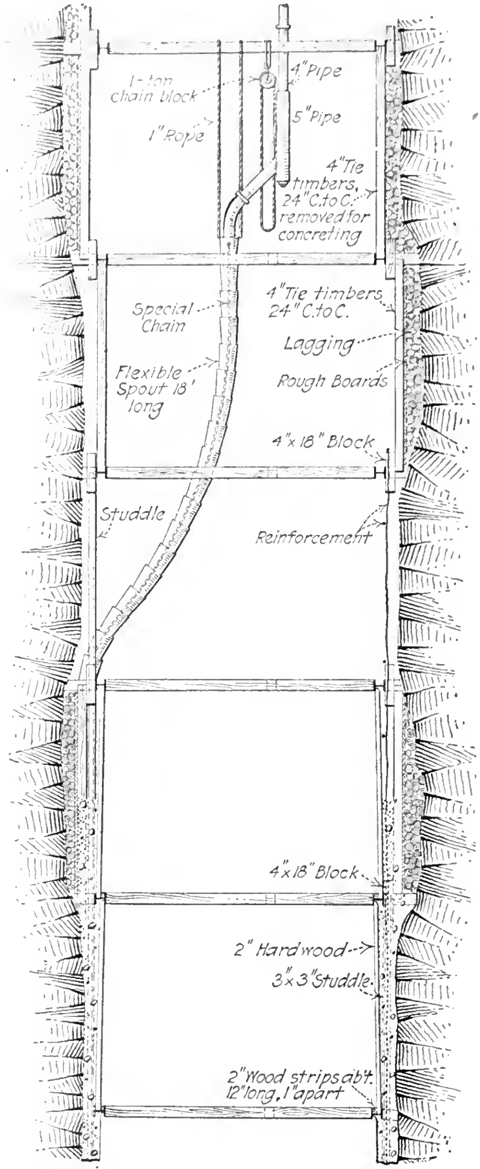


FIG. 98.—VERTICAL SECTION OF SHAFT, SHOWING SETS AND FORMS AND FLEXIBLE SPOUT.

to the forms. On the surface, three men wheeled rock to the mixer, two men the sand and cement, one poured water and attended to the securing of the proper mixture, one discharged the mixer, one looked after the launder from the mixer to the 4-in. pipe, and two men worked the concrete down the 4-in. pipe. All the men worked 8-hr. shifts on the concreting.

The speed of sinking the shaft, including the placing of steel sets and lagging, occasional concreting, etc., averaged from 4 to 4.56 ft. per day during several months. For the last three weeks in August, 1913, it averaged 5 ft. per day. The speed of final concreting was from 35 to 48 ft. per day. For the total distance concreted, 78 gondolas of sand and 15,695 sacks, or 21 carloads, of cement were required.

Unit Sets of Reinforced Concrete (From a paper by E. R. Jones before the Michigan College of Mines Club).—Concrete members molded separately on the surface were made for the No. 3 and No. 4 shafts of the Ahmeek Mining Co. The shafts are sunk at an angle of 80° and are 3-compartment shafts, two skipways and one manway. The outside dimensions of the compartments are: skipways, 7 ft. 6 in. high, by 6 ft. 10 in. wide; manway, 7 ft. 6 in. high, 3 ft. wide, with the end plates and dividers making the greatest span of 7 ft. 6 in. Offsets were molded in all plates 5 in. from the inside face to accommodate lining slabs. Also, holes were cored for hanging and bracket bolts. The wall plates, end plates and studdles had a cross-section of 80 sq. in., the dividers 81 sq. in. The percentages of reinforcement were approximately as follows: Wall and end plates and dividers, 5 per cent.; studdles, 3 per cent.

The materials finally selected were portland cement, conglomerate sand, and trap rock trommeled over ¾-in. screens. The proportions were 1:3:5 in wall plates, end plates and dividers, and 1:2:4 in studdles. The reinforcement in wall and end plates consisted of three ¾-in. monolith steel bars with ¼-in. webs crimped on, together with two straight ¾-in. monolith bars. The dividers were reinforced by four ½-in. monolith steel bars wound spirally with ¼-in. steel wire, the whole presenting a column of square cross-section. Studdles were reinforced with two pieces of old wire rope 1¼-in. in diameter. Slabs were molded for the shaft lining, the materials used being trap rock under ¾ in., conglomerate sand and Kahn expanded metal as reinforcement; the mixture was 1:2:4.

In molding, 2-in., No. 1 white pine was used for the forms. The forms were soaked in Delaney's wood preservative and repainted with preservative on the interior each time before setting up, thus insuring them against warping and prolonging their lives indefinitely, as well as securing a smooth and easy parting of the concrete when removed.

The labor involved in making the sets consisted of two carpenters setting up forms and keeping them in repair; one man wheeling forms

on to skipways ready for filling, and returning used forms to the shop and cleaning them; one man feeding mixer from stockpiles of rock, sand and cement; one man delivering mix to forms and shoveling material into place; and one mason ramming charge into final positions. With this combination of men as many as four complete sets, consisting of 64 separate pieces, have been molded in one day of 9 hr. In ordinary weather, the sides of the forms were allowed to remain in position over night and then removed, while the bottoms were left in place another 24 hr. The bottoms were removed by turning the pieces on their sides where they were left to harden one day longer before removal to the stockpile. All through the process of removal the sets were handled with the greatest care in order to preserve the appearance of the set and prevent cracking, which might not develop so as to be visible to the eye until weathered. All skidways used in making and storing were brought to a level to prevent warping and bending while the sets were green, in order to insure a perfect set under ground, for, unlike timber, concrete sets cannot be brought to place unless perfectly true. Sets should not have been used under 60 days after removing forms, although, through the reduction of the stockpiles, it was at times necessary to install pieces of 14-day sets; the greatest care, however, was observed in handling and putting these in place underground. Concrete sets one year old, which had been subjected to all kinds of weather, can be abused somewhat and handled almost as carelessly as timber.

It was found advisable from the beginning, because of the great weight of the wall plates, to mold them in two sections, one section spanning the ladderway and one skipway, and the other section spanning the remaining skipway. These two sections were connected when in place by two bolts passing through holes cored for the purpose and two straps of iron spanning the splice. Studdles were made for 4-, 5- and 6-ft. sets, to accommodate the ground passed through.

The weights of the different pieces comprising the sets were as follows: Long section of wall plate, 1035 lb.; short section of wall plate, 700 lb.; end plate, 600 lb.; divider, 645 lb.; and 3 × 3-in. studdles, 268 lb.; a total weight of 8104 lb. for a complete set of 16 pieces.

Taking the weight of No. 1 western fir which has been exposed to the weather in stockpiles, as 33 lb. per cubic foot, the concrete set weighed almost three times as much as a 12 × 12-in. timber set which the concrete set was intended to replace. Because of this additional weight of the concrete set, it was found necessary to increase the usual five or six men on the timber gang to seven. In a vertical shaft, to which the concrete sets are especially adapted, the number of men per gang might be reduced. The sets were hung or built as the ordinary timber sets, only requiring an additional rope and block with which to swing the pieces

into place. After the sets were wedged to line, bottoms were put in between the plates and the surrounding shaft wall, and the set then tied to the shaft by means of concrete in the proportion of 1:3:5. The concrete slabs were next put in place and loose rock thrown behind them, filling up what space still remained between the set and the wall of the shaft.

After the set was in place, it was extremely important that it be well protected from the blasting. For this purpose there were used flat timber and steel plates chained to the under side of the plates and dividers, and even this precaution was at times inadequate. Where the ground was breaking easily, the sets have been as near as 12 ft. to the miners, and again, where the ground was especially hard and tough, sets 40 ft. from the blast have been cut out. In dangerous ground, which required timbering close up to the sinking, timber sets were used, but had not time played an important part in the sinking, no ground was met in which concrete sets could not have been installed. With a gang of seven men, a complete set can be installed in a 9-hr. shift. This permits a sinking rate of more than 100 ft. per month, which was accomplished at the two shafts.

The comparative costs of the concrete and timber sets delivered at the shaft collar is striking. The concrete set was delivered for $22.50, the timber set for $37.60. These figures are based on western fir at $28 per 1000 ft., f.o.b. car; crushed rock at 35 cts. per cubic yard, f.o.b. shaft; conglomerate sand at 60 cts. per cubic yard, f.o.b. shaft; No. 1 portland cement at $1.15 per barrel, f.o.b. works; reinforcement at $12 per set, f.o.b. factory.

Relining the Hamilton Shaft (Lake Superior Mining Institute).—The No. 2 Hamilton shaft of the Chapin mine at Iron Mountain, Mich., was sunk in 1891, and consisted originally of six compartments as shown in Fig. 99, two for skips or bailers, 4 ft. 8 in. by 7 ft.; two for cages, 4 ft. 8 in. by 4 ft. 6 in.; and two for pipes, etc., 4 ft. 8 in. by 2 ft. 0 in., taken off the ends of the cage compartments. This shaft was timbered with 16 × 16-in. material in the main members, the sets spaced 6 ft. 2½ in. center to center, the outside lagged with 2-in. planks, making a minimum rock section of 10 ft. by 24 ft. 4 in.

The timbers, due to long service, became badly decayed, so that it was necessary to reline the shaft or abandon it. Early in 1911 it was decided to make it a permanent outlet for the Chapin mine, and install in it the permanent electrical centrifugal-pumping equipment. It was, therefore, necessary to reline it from collar to bottom, a distance of 1434 ft., and since there was a possibility of striking a heavy flow of water in the underground workings at any time, provision had to be made so that bailers could be put in service on short notice. Since it was to be the

permanent outlet, further provision had to be made for column pipes and electric cables. To provide for these pipes and cables, it was necessary to increase the inside dimensions of the shaft from 7 by 21 ft. 4 in. to 9 by 21 ft. 4 in., making a poured concrete wall 6 in. thick. The outside dimensions of the shaft were not changed. In the new design, the shaft consists of eight compartments: Two for skips or bailers and two for cages, each 4 ft. 8 in. by 6 ft. 4 in.; three for pipes and cable and one for ladder, each 2 ft. 4 in. by 4 ft. 8 in., the skip compartments set off with concrete-slab partitions. The arrangement is shown in Fig. 100.

To make the concrete economically, a mixing plant was built near the shaft. The dividers, end plates and slabs were made in steel forms, placed beneath the mixers so that the concrete could be poured directly into them. After the concrete had sufficient time to harden, the forms were removed and the molds picked up by a hand-traveling crane and carried into a drying room, where they were cured. The design of these members is shown in Fig. 100.

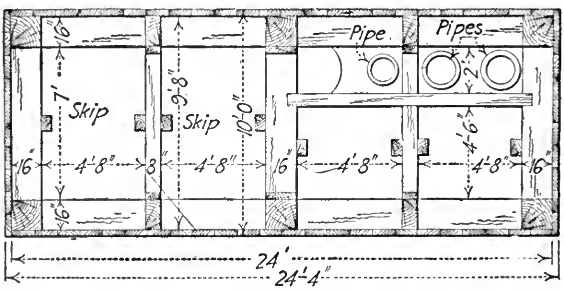


FIG. 99.—PLAN OF OLD SHAFT LINING.

The actual work of relining was done in sections, each section being carried upward from permanent bearers which were set to support the old timber sets. A set or two of old timber, usually 12 ft., was removed, loaded on the cages and hoisted to the surface. The timber sets above the removed portions were supported by vertical columns with jack screws on the bottom, which rested on 12 × 12-in. timber placed on the bearers. After the first 6-ft section of concrete was poured, the 12 × 12-in. timbers were placed on the reinforced-concrete dividers and end plates, which were themselves supported on the steel wall-forms, as shown in Fig. 101. The steel forms were made in sections, with recesses to support end plates and dividers, and spaced on either 4-ft. or 6-ft. centers. The original forms were designed for 4-ft. centers; when the 6-ft. spacing was used, a section was bolted to the top of the form. Since there were seven sets of steel forms, the footings to carry the weight of old timber sets bore either on the permanent bearers or on at least five sets, 30 ft., of concrete, *i.e.*, the support of the old timbers above did not depend upon green concrete.

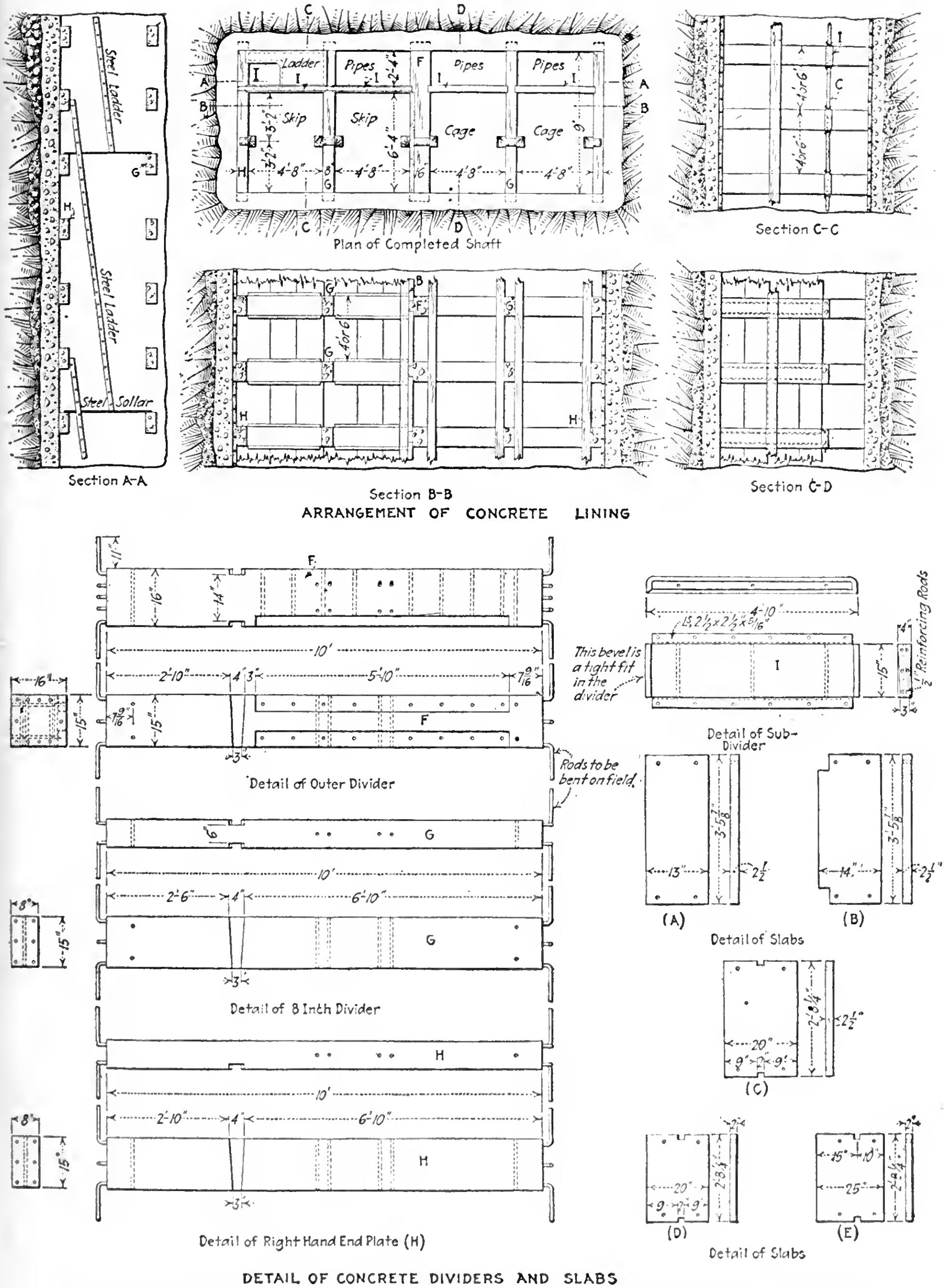


FIG. 100.—NEW CONCRETE LINING OF HAMILTON SHAFT, IN DETAIL AND ASSEMBLED.

After a new set of steel forms was lowered in the cages and installed, the dividers and end plates were lowered and placed in the recesses provided in the steel forms, with their ends bolted to the forms. These end plates and dividers served as horizontal struts to hold the forms in position. When a section was placed, the vertical reinforcing rods were put in position and the wall was ready to be poured. The dividers and end plates had projecting reinforcement rods for anchoring the wall.

The concrete for the wall was mixed in the surface mixing plant and discharged into side-dump steel cars, which were hand trammed to the shaft. A turntable was installed so as to serve both skip compartments. The concrete car was run on a cage and lowered into the shaft. A revolv-

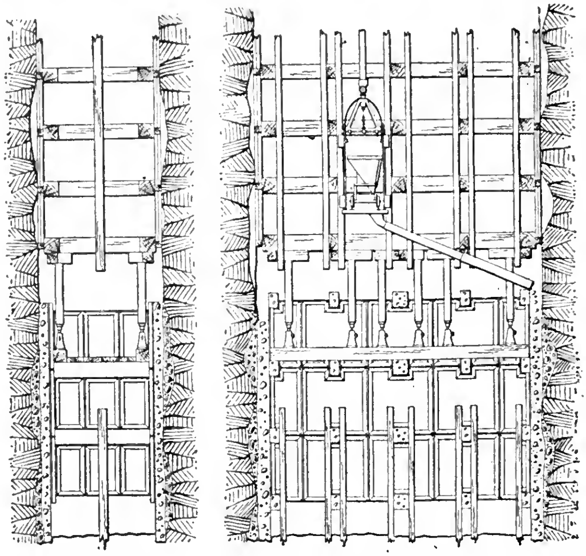


FIG. 101.—METHOD OF INSTALLING LINING, HAMILTON SHAFT.

ing chute was attached to the spout of the car and the contents were discharged behind the forms and properly tamped in place. Where large crevices occurred, they were filled to within 10 in. of the steel forms with large stones or rock from the over-size bin before the concrete was poured. The average amount of material for relining one 6-ft. vertical section of shaft was 1 cord of stone for backfilling, 10 cu. yd. of concrete, and 550 lb. of steel for reinforcing. The slabs for the skipway partitions were bolted to the dividers.

In removing old timber, five men were required below and one man at the collar. The time required to remove one 6-ft. section varied according to the condition of the material. In placing steel forms or pouring concrete, four men were required below and two at the collar. This

does not include the shaft foreman or the concrete foreman, as these two men did not spend all their time on this particular job. The work was carried on with three 8-hr. shifts and the average time required to concrete 6 ft. of shaft was 24 hr., including placing the forms, pouring concrete and removing an equal number of forms from below. When the forms were removed they were taken to the surface, thoroughly cleaned and given a coat of crude oil before they were used again.

The first section of relining was started on May 3, 1912, 83 ft. 3 in. below the collar, and the lining between this point and the collar was completed June 29. The fifth section was started at 917 ft. May 17, 1913, and on July 5, 813 ft. of shaft was completed. The average rate of progress over this time, without deducting the time due to delays, was 56.7 ft. per month and 63 ft. per month for actual working time. The progress for the last month was 72 ft. The preliminary estimate was based on relining 100 ft. per month. The old shaft timbers, however, were in far worse condition than could possibly be anticipated, and the slower progress was due entirely to the difficulty in removing them and the precautions required to protect the lives of men employed on this work.

In the portion of shaft completed to the date of writing, all the work proved perfectly satisfactory and entirely up to expectations. The walls were smooth and waterproof. The reinforced-concrete dividers and end plates came from the steel forms perfectly true, straight and smooth, and fitted perfectly in the recesses provided in the steel forms.

Unit Concrete Sets from Central Factory (By L. D. Davenport).—The Oliver Iron Mining Co. erected a concrete plant at Hibbing, Minn., for making building blocks, shaft sets, slabs and similar products for use at its new mines on the Mesabi range. The structure is a frame building covered with corrugated sheet iron. It is 115 ft. long over all and consists of a main plant, 24 × 57 ft., four stories high, and a one-story shed 46 × 91 ft., divided into drying and steam rooms.

The sand and rock for the concrete are brought in stripping cars from one of the openpit mines and dumped close to the plant. This material as needed is trammed in a small car over a low trestle to the crusher. From the crusher a belt conveyor carries the crushed material to a trommel at the top of the plant, where it is sized and deposited in three bins as shown in the flow sheet. A 30-hp., 200-volt, Westinghouse motor runs the crusher, the trommel and the conveyor belt. The cement is brought to the plant in railroad cars over a trestle at the elevation of the charging floor and is unloaded and stored under the bins conveniently near the charging car. Below the bins is a track carrying a wedge-shaped charging car which is divided into compartments for sand, rock and cement. The charging car dumps directly into a No. 1

Smith mixer, water being added from a tap close by. This charging car is also used to tram the oversize from the screens to a chute leading back to the crusher. From the mixer the concrete is taken to one of three chutes by a transfer car of the same pattern as the charging car except that it is not divided into compartments. One of the chutes feeds a Denver block machine and the other two supply the various forms placed below them. All blocks are cured in the steam room and the shaft sets are either sent to the steam room or are dried in the main shed, which is steam heated. The finished product is taken from the trucks and piled or loaded on freight cars by means of an air lift of about 5

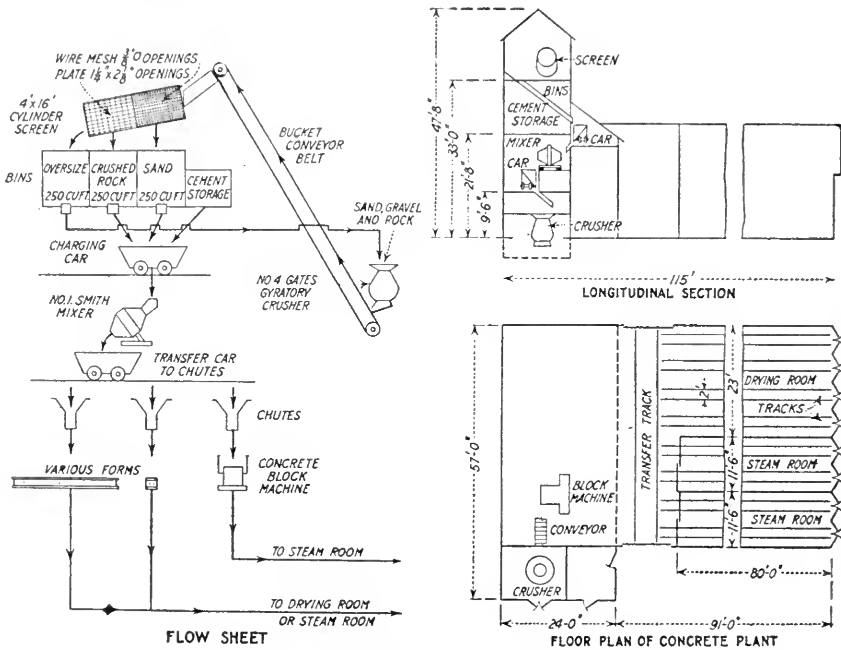


FIG. 102.—HANDLING MATERIAL AT MANUFACTURING PLANT.

tons' capacity, running on an overhead crane. The arrangement of the building and the successive steps in handling the material are shown in Fig. 102.

A 1:4 mixture of cement and sand is used in the blocks and a 1:2:4 or a 1:2:3 in the shaft sets. Three complete shaft sets, *i.e.*, six wall plates, six end plates, six dividers, 24 studdles and 156 slabs for lagging, and from 500 to 700 9 × 24-in. building blocks constitute an average day's output under favorable conditions.

The concrete shaft sets are cast in forms built up of steel plates and angles with all holes cored tapering about 1/8 in. larger than the bolts to

be used. A 1:2:4 mixture was used at first, but this was changed to a 1:2:3 in order to obtain a smoother product. The concrete is made wet and puddled in the forms rather than rammed. The wall plates and end plates are 12 × 12 in., and the dividers are 10 × 12 in. The corner studdles are 10 × 10 in. and the center studdles are 8 × 10 in. The studdles being 4 ft. long, the sets are spaced on 5-ft. centers. The general style of framing is the same as for the standard wood-lined shaft 6 × 18 ft. inside, with the exception of a few details as follows: A 2 × 2-in. cleat spiked in the center of the outside of the wooden sets makes a bearing for the lagging, while the upper and lower outside edges of the concrete sets are rabbeted 1½ × 3 in. to hold the concrete lining slabs. The details of the members and the arrangement at the collar are shown in Fig. 103.

The shaft is hung from four steel crossbearers which in turn rest on two reinforced-concrete longitudinal bearers about 2 × 3 ft. in cross-section and 34 ft. long. These were cast in place just below the surface of the ground from the outside of the shaft. The crossbearers at each end of the shaft are each made up of two 15-in., 33-lb. channels 19 ft. long. Those at the dividers are single 15-in., 42-lb. I-beams of the same length. The shaft collar including all the shaft above the first bearers was cast in one piece with 12-in. outside walls and 10-in. dividers. The placing of the first two or three sets directly below the bearers presented a slightly different problem from the ordinary timbered shaft on account of the solid concrete collar section above the bearers. In a timbered shaft the dividers are left out of the collar set until the next two or three sets below have been hung. With a 6- or 8-ft. monolithic collar section including the dividers, there is not room to lower and swing into place the 20-ft. wall plates for the next two or three sets except by sinking 18 or 20 ft. without timber. This difficulty was overcome by driving a 2 × 3-ft. flat incline from the surface to a point just under the crossbearer at one end of the shaft. The concrete timbers were slid down this incline and hung in place until the shaft was sufficiently deep to allow lowering in the ordinary manner.

The skip guides are of wood and are fastened to the end pieces with 3½ × 7/16 × 7-in. angles and to the dividers by 7½-in. lengths of 10-in., 25-lb. channels. All ladderway sollars are 5/16-in. hob steel plates with steel toe-boards of 2 × 2 × 3/8-in. angles around the ladder openings and of 5 × 3½ × 3/8-in. angles on the pipeway side. The ladders are made of 1½ × 1½ × ½-in. angles with ¾-in. pipe rungs spaced 10-in. centers. Two pieces of 1¼ × 1¼-in. flat iron cover the ends of the rungs and are held by 3/8-in. bolts through the rungs and angles. The ladders are about 24 ft. long and extend 4 ft. through the sollars to the under sides of the sets above, making excellent hand holds. The ladder-

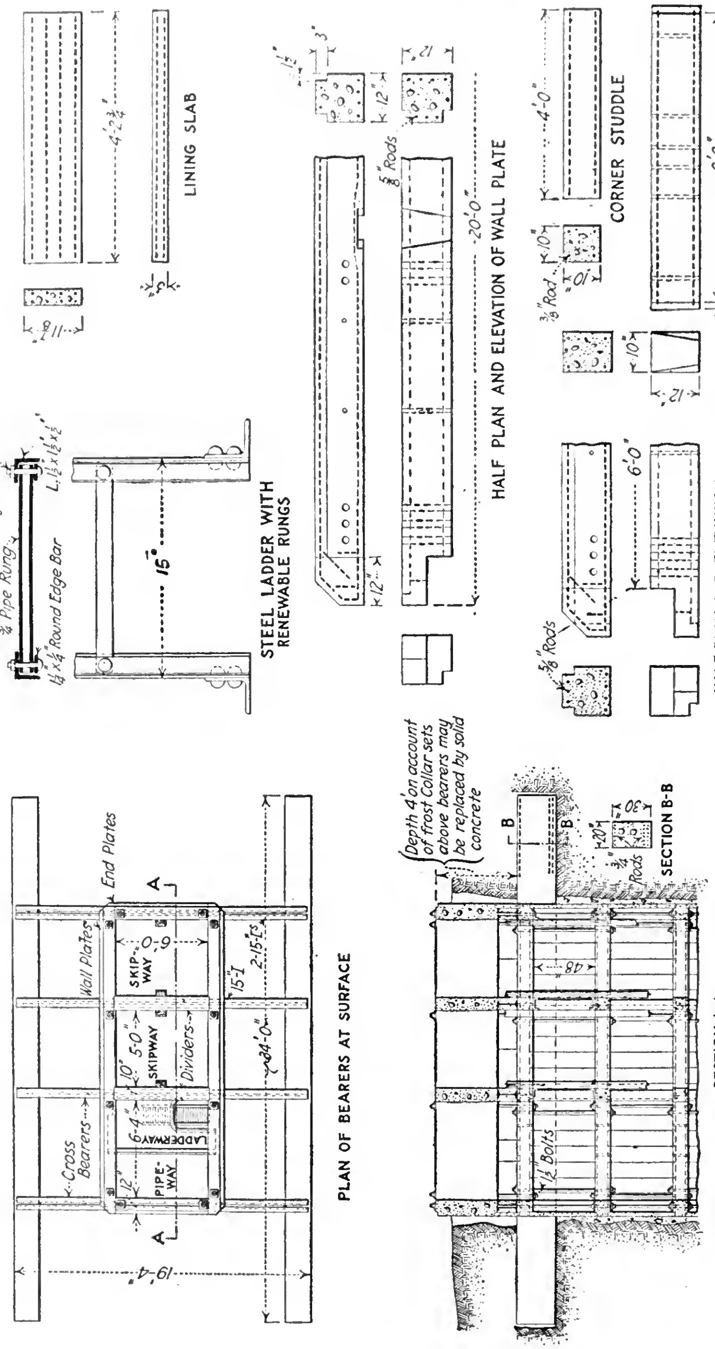


FIG. 103.—SHAFT ARRANGEMENT AND DETAILS OF SHAFT MEMBERS.

way is separated from the skipway with a No. 26A galvanized reinforcement mesh.

Gunite Casing on Wood Shaft Lining (By Stephen Royce).—An experiment tried at "A" shaft of the Cary mine, operated by Pickands, Mather & Co. at Hurley, Wis., on the Gogebic range, consisted of covering the steel sets and wood lath with cement mortar applied with a cement-gun. The shaft arrangement is shown in cross-section in Fig. 104. The inclination is 73° 10′. The steel sets are spaced at 4-ft. intervals and the 3-in. hemlock lagging is wedged into the flanges of the I-beams, as shown. The sets were blocked into place by wooden blocking, with

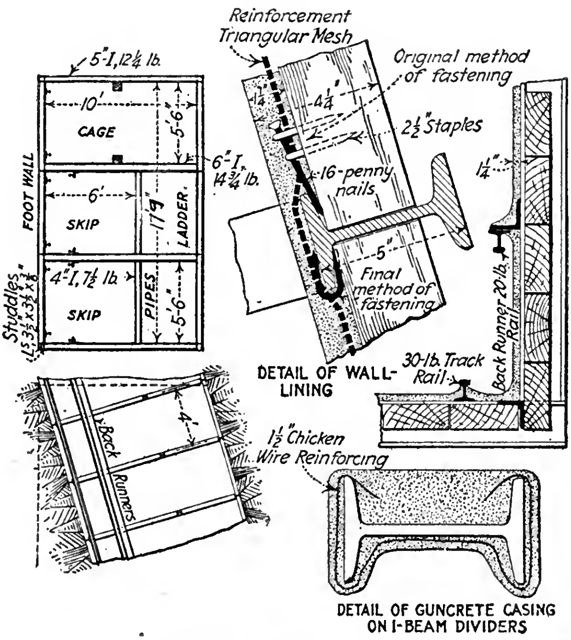


FIG. 104.—SHAFT ARRANGEMENT AND DETAILS OF COATED MEMBERS.

bearers put in at intervals of about 50 ft. At the several level openings the shaft was completely concreted before the cement-gun work was contemplated. The total depth of the shaft is 1290 ft.

The hemlock lagging showed signs of rotting out and of course it permitted free circulation of air behind the sets, which it was feared would rot the blocking and so throw the shaft out of line. In order to fire-proof the wooden lining, to protect the steel work from rust, to exclude air from the spaces between the sets and the outside rock, and to keep all the wood uniformly wet so as to discourage decay, a coating of 1¼-in. "Gunite," as the material applied by the cement-gun process

is called, was decided upon. This has been tried in two experimental sections of the shaft having a total length of 263 ft.

The cement-gun is an apparatus designed to shoot through a nozzle a stream of mixed cement, sand and water in about 4 :1 mixture. One of the advantages claimed for mortar shot on by this process is that the mixture is automatically enriched where most needed and is less rich where a poorer mixture is sufficient. This results from the tendency of the particles of sand to rebound as soon as they hit a hard surface, so that the first 1/8 or 1/4 in. will consist of almost neat cement. This material sets almost as soon as it strikes and the full benefit of the set is given to the work, there being no possibility that the cement may partially set between the time of mixing and the time of applying.

The delivery hose is made of pure soft rubber, covered with heavy canvas and may be as much as 300 ft. or more in length. The wearing out of the delivery hose is one of the chief expenses of operating; one hose may last for from two to six weeks, according to the character of the sand that is fed.

Of the two sections of shaft covered in this work, one extended from the collar to the 3rd level, the other from the 8th to the 10th level. In the upper section the machine was placed on the surface and the delivery hose was carried down through the shaft to the point where work was being done. In the lower section the machine was set on the 8th level. Material and supplies were handled in a temporary cage in the center compartment, no hoisting being done in the shaft at this time.

The walls were reinforced with No. 7 American Steel & Wire Co. triangular mesh and the dividers with 1½-in. chicken wire, held on with wire clamps. Fig. 104 shows two methods of attachment of the reinforcing wire to the steel sets. The first method used nails and staples. This was discarded because it placed too much dependence upon the holding power of the staples in the hemlock planks, which might later decay. The second method depends entirely upon the strength of the staples, which hold directly on the steel, and was adopted for most of the work. The reinforcing wire was also stapled to the lagging at intervals between the sets. The dividers were filled in completely on their upper faces, the result being a considerable increase in their ability to resist a downward load. This increase of strength is figured at nearly 20 per cent.

The results seem to have been entirely favorable. There is a solid hard coating of concrete over the entire sides and all the steel work in the shaft. This coating seems to be completely waterproof, so much so in fact that it was considered necessary to set pipes in the bottom of each section coated in order to prevent the water from banking up behind the shaft covering and causing heavy hydrostatic pressure. What

the effect will be upon the life of the lagging and of the blocking behind the sets is not known, of course, as yet, but from the uniform and apparently airtight character of the coating, it is hoped that it will greatly increase the life of the wood. If the lagging partially rots, the reinforcement is so solidly fastened to the steel that it should take up a considerable part of the stress now borne by the lagging.

Concreting Methods in Copper Range Shafts.—In the Baltic mine the concrete support for the roof of the shaft is reinforced with $1\frac{1}{4}$-in. ropes, three carried down each skipway and stretched between eye-bolt anchorages. The ropes are stretched until at the center they are about 6 in. from the edge of the concrete. At intervals of 8 to 14 in., cross-reinforcement of the same material is put in. Alternate ropes of this cross-reinforcement are turned up about one-third of the way out from

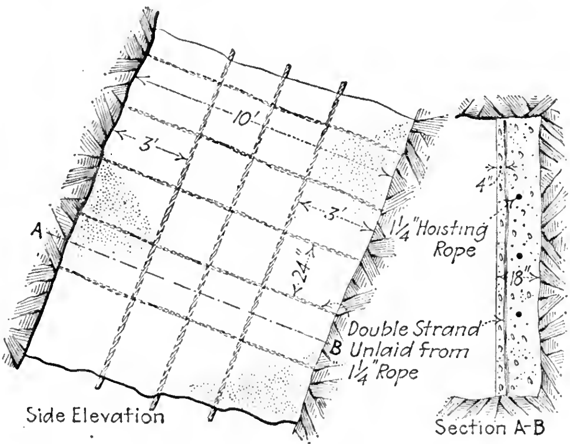


FIG. 105.—ROPE REINFORCEMENT IN SIDE LINING.

the side walls and also turned up to pass over the dividers for the purpose of taking the shear.

Along the sides, three ropes in the direction of the shaft afford reinforcement. Cross-reinforcement in these side walls consists of double strands unlaid from $1\frac{1}{4}$-in. cable. These are set in a vertical plane and at right angles to the long axis of the shaft, about 2 ft. apart and 4 in. from the face of the concrete. Fig. 105 shows the general arrangement.

When there is continuous concrete on the hanging, dividers are put in 10 in. thick and 6 ft. along the shaft with 6-ft. spaces between. These are shown in Fig. 106. Three $1\frac{1}{4}$-in. ropes in the direction of the shaft and three pairs at right angles, spaced as shown, afford reinforcement. The pair lowest in the divider is bent at the ends to take shear. Bolts in the direction of the shaft axis are set in at top and bottom of the divider and serve to carry 3-in. nailing strips to which the planks covering the

spaces between the dividers are nailed. These nailing strips are set back 2 in. so that the planks are flush with the concrete and are thus protected against being knocked off by falling rock.

In case the hanging wall stands but the sides give trouble, the latter are reinforced as described and dividers also set, but with the spaces between increased to 10 ft. To take the side thrust of the walls, a cross-beam against the hanging is put in about every 25 ft. This is 10 to 18 in. thick and 4 ft. wide as shown in Fig. 107. This brace is reinforced with three 1¼-in. ropes across the shaft. It has its top face beveled to prevent material from lodging. These beams are set at the same points as the dividers so as to be braced against side bending.

The stations are equipped with concrete brow-pieces over the skip-

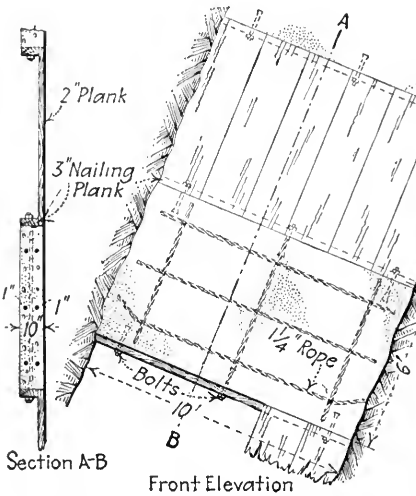


FIG. 106.—CONCRETE AND PLANK DIVIDER.

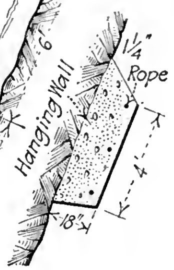


FIG. 107.—HANGING-WALL CROSSBEAM.

ways, shown in Fig. 108. These are reinforced with a grilling of double strands across and single strands in the direction of the shaft, so put in as to form 6-in. squares. The angle at the floor of the station is strengthened with a 55-lb. rail.

In the Champion shafts, the roof reinforcement of 1¼-in. rope is put in as a network in the manner shown in Fig. 109. The cross-ropes are strung through eye-bolts and carried up and across in a zigzag manner. The ropes in the direction of the shaft are anchored only at the top and bottom. The ropes in the network are clamped at the crossing points and ends. The approximate spacing is as shown.

Concrete Collar for Mohawk Shaft.—Incline shafts of the Lake Superior copper district are now commonly equipped with a concrete shaft

collar to keep out surface water and to keep fires from extending underground. Generally the shafts have to be sunk through a considerable depth of overburden, often a sort of quicksand. The general procedure is to strip a large enough area of overburden to open a pit in which the sides stand at the angle of repose of the material without interfering with

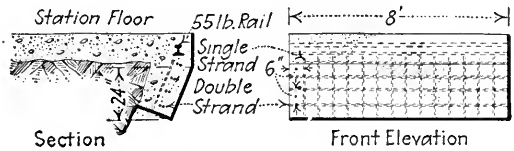


FIG. 108.—CONCRETE STATION BROW REINFORCED.

the putting in of the shaft collar. The concrete collar is started from a bench cut down to solid rock so as to permit making a water-tight joint between the rock and the collar. After the collar is finished, the ground is piled in again to fill the excavation.

At Mohawk No. 6 shaft, the collar was 99 ft. long with outside dimensions of 22 × 11 ft. and walls 12 in. thick; the overburden was 45 ft. deep.

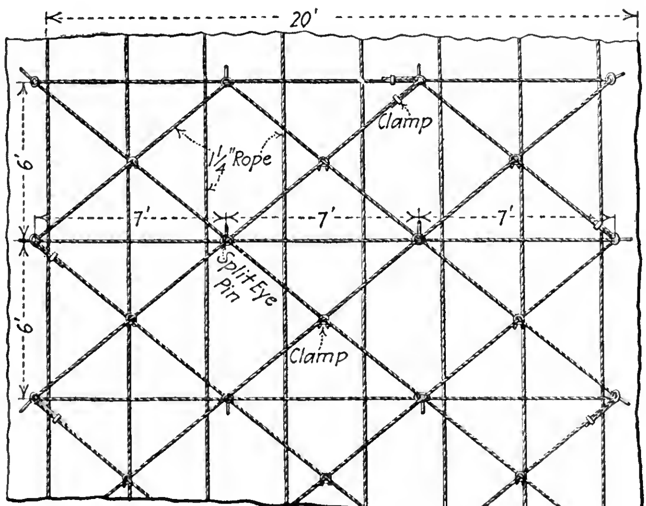


FIG. 109.—ROPE REINFORCEMENT ON HANGING WALL.

The concrete was poured down troughs from a mixer at the top. In order to keep the concrete from separating as it went down, splash boards were put in the troughs every few feet to retard the descent. At the bottom the concrete was caught on a platform, turned over with a shovel and then shoveled into the forms. As the work progressed upward, the platform was raised. A 1 :3 :5 mixture was used, the aggregate being rock

from the mine, crushed to pass a 2-in. screen, with the fines left in. Knockdown forms of wood were used, being moved up as concreting progressed. Fig. 110 shows the manner of building the forms, and the way that they were held in place by braces and bolts on the inside as well as on the outside.

For reinforcing, discarded 1⅛-in. hoisting cables were used. This rope was put in to form the cross-rods for strengthening the bottom, top and sides of the collar, and each alternate rope was turned up at the ends at an angle of 45° to take care of the shearing stresses near the ends. The rope was assumed to have half the strength of an iron bar of the same area, and the area of the rope reinforcement at the bottom of the collar was kept at about 2 per cent. of the sectional area of the walls. The ropes were spaced at intervals of 6 in. in the lower part, but the spacing was increased toward the top until it was about 10 in. The ropes were placed in the forms so that they would be about an inch from the inside surface of the walls, approximately one-tenth the depth of the wall.

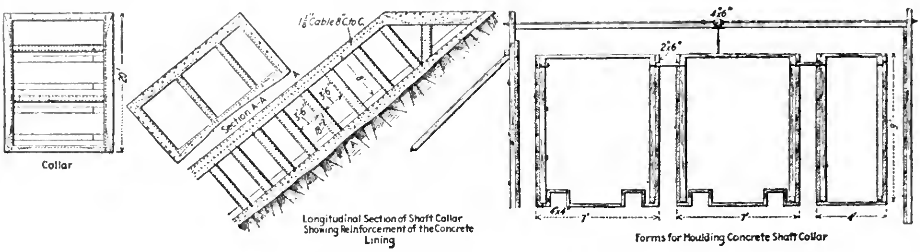


FIG. 110.—CONCRETE COLLAR OF THE MOHAWK SHAFT.

The divider columns were 12 × 18 in. in section reinforced by four ¾-in. bars wound with ¼-in. rods to hold them in position with respect to one another. The rods for strengthening the dividers were fastened to two pieces of hoisting cable that ran down along each line of dividers, and reinforced a beam, 6 in. deep, which connected the dividers.

Heavy poultry netting was put in the collar as centering to hold the pieces of rope in their proper relation to one another and to the forms; to this the rope was wired. After this netting and the ropes had been nailed to the inside forms, the outside forms were erected around it, and the concrete was shoveled in. The planks of the top form were nailed on as fast as the forms were filled with concrete. After about four forms had been filled, and the concrete in the bottom form had set about a day and a half, the outside forms of the bottom section were removed by loosening the top bolt and knocking out the side braces, while the inside forms for the walls and the columns were also taken off. The forms were then reërected and used again.

The 100 ft. of collar was completed in 17 days after the beginning

of concrete work. In its construction 9000 ft. of 1⅛-in. hoisting cable weighing 9 tons was used together with 176 rods ¾-in. in diameter and 11 ft. long, weighing about 1½ tons; 888 ft. of ¼-in. rod weighing 147 lb., 4500 sq. ft. of heavy poultry netting; 16, 45-lb. rails; two 60-lb. rails and 12,000 bd. ft. of pine. The rails were used in reinforcing the bottom of the collar where it was seated on the ground. The total cost was $3931.

Concrete Shaft Collar at the Wolverine.—At the Wolverine No. 5 shaft, sunk to prospect the Osceola lode, a concrete collar was put in 56 ft. long and 22 × 11 ft. in section; outside measurements. The reinforcement was applied in a manner different from that used at the Mohawk No. 6, partly because of less depth of overburden, only 16 ft. Triangular-mesh reinforcement was used to carry the ropes in the forms as well as to aid in reinforcing the collar.

In prospecting for the lode it was necessary to trench through the overburden. This trenching dried up the wells, as the shaft is in a small

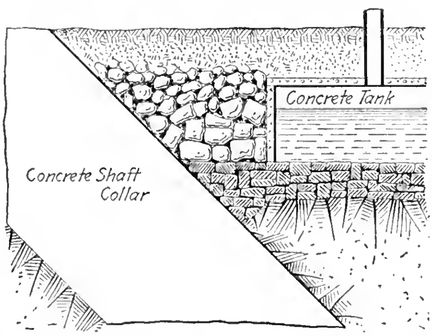


FIG. 111.—CISTERN FOR COLLECTING BOILER WATER ABOVE CONCRETE COLLAR.

gully draining the country near-by. In order to restore the wells to use and to enable the making of a well for supplying the boiler plant, it was decided to make the collar also serve as a dam to back up the water. A funnel was put through the concrete to let the water into the shaft while the collar was being erected. Then, after the collar had been completed, the surface that was to be toward the water was coated with tar to help make it waterproof. To make a cistern for the boilers, a dry wall of rough masonry was put in on the bottom of the shaft trench, as shown in Fig. 111, and a tank with concrete walls was erected on that as a foundation. This tank was covered with concrete, a terra-cotta tile 24 in. in diameter being inserted in the top to serve as a manhole. After this tank had been completed a piece of waste saturated with cement was inserted in the funnel and rammed tight. This shut off the flow of water into the shaft almost instantly, and concrete was then poured in on top of the waste. Next, large boulders were piled in the pit around the shaft and

tank, and finally dirt was put on top, and the concrete cistern completely covered over with dirt.

The water rose in the cistern and reserve for the boilers was obtained, as the water would filter through the loose rock around the collar and up through the loose masonry at the bottom of the tank into the cistern as fast as it was pumped from it. About 16,000 gal. of water was thus stored back of the dam, and the water rose to a height of about 12 ft. above the place where the collar was seated on solid ground; little seepage resulted.

Owing to the fact that triangular-mesh reinforcement was used to center the rope reinforcement in the forms, and that the depth of burden was only 16 ft., the pieces of old hoisting cable were put in 7 in. apart at the bottom. As the collar grew in height the interval was increased to 10 in. Only at the bottom were the side walls reinforced. In other respects the reinforcement was similar to that used in the Mohawk; the knock-down forms and the delivery of the concrete by sliding it down troughs to a plat whence it was shoveled into the forms, were adhered to.

Concrete Drop Shaft (By Claude T. Rice).—The shaft for opening the newly discovered lode of the Indiana Mining Co., in the Lake Superior copper region, had to be sunk through 100 ft. of overburden, in the upper 60 ft. of which there was sand and clay, followed for about 20 ft. by material in which quicksand and clay prevailed and in which were one or two seams of sand and gravel about 1 ft. thick. From 70 to 77 ft. from the surface, the material was hardpan succeeded by sand, clay, and quicksand, which extended to hard rock. This overburden was penetrated by a concrete drop shaft. The drop-shaft casing was made of annular sections of steel plate, 18 ft. in diameter at the bottom, but with each ring of slightly smaller diameter, so that each one bolted inside the one below. The sections were overlapped 4 in. The two lowermost rings were of $\frac{1}{2}$-in. plate; those above were $\frac{3}{16}$ in. thick. A $\frac{5}{8} \times 6$-in. angle was fastened to the bottom plate to stiffen the bottom and give a somewhat stronger cutting edge. This angle-iron ring, furthermore, facilitated the concreting that was to follow.

The first 50 ft. of the shaft was sunk in winter without any trouble being experienced, as the ground froze and stood without support. The thickness of the concrete shell was proportioned to give weight enough to overcome a frictional resistance of 400 lb. per square foot. No allowance, however, had been made for the fact that the 50 ft. of sand that stood without support would offer little resistance to the settling, and, as a result, the drop-shaft sank down through the overburden faster than the sand could be excavated, causing no difficulty other than that the cutting edge was bent up somewhat by forcing boulders out of its path.

The concrete lining of the steel ring was put in from the bottom up; in the first 50 ft., collapsible wooden forms were used. The shaft opening

within the lining was octagonal; 10-in. channels are shown in the center of the octagon, Fig. 112. These were put in at 6-ft. centers, to carry the center guides, which in this concreted part of the shaft are 6-in. channels. The end guides are secured by brackets set in the concrete. These brackets were not put in until after the drop-shaft had been seated upon solid rock. Instead, blocks of wood were put in the concrete at the places where the brackets were afterward to be set, this being necessary because the drop-shaft was continuously sinking and the hoisting and lowering of material was done by a boom derrick. As this bucket could not be

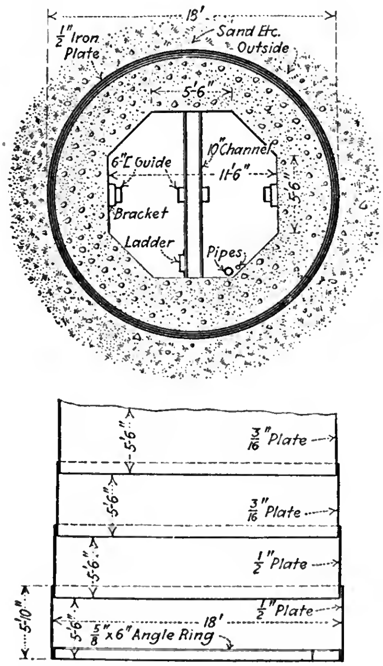


FIG. 112.—CONCRETE DROP SHAFT, PLAN AND SECTION.

used with guides, had the brackets been set in place at first, the buckets would have caught upon them.

After the concreting in the first 50-ft. section of the drop-shaft had been carried up to the surface, further steel plates were bolted to the top of the completed portion; the collapsible wooden form was moved up, and a new section concreted, the operation being repeated as rapidly as the drop-shaft sank. Reinforcement was used only in the upper 10 or 12 ft. of the completed drop-shaft, consisting of a few wire ropes, in 1:3½:7 concrete. The concrete for the lower part of the shaft was a 1:2½:5 mixture.

When solid rock was reached, the material remaining within this

shaft was excavated; a smooth shelf was cut in the solid rock by hammer and drill; and the joint was sealed with concrete. Work was started Jan. 1, 1911, and the drop shaft completed about Apr. 1; the total cost was $57.58 per foot.

CONCRETE SKIP STRINGERS

Concrete Stringers for Incline Tracks (By Claude T. Rice).—In the Michigan copper district the use of concrete stringers under the skip

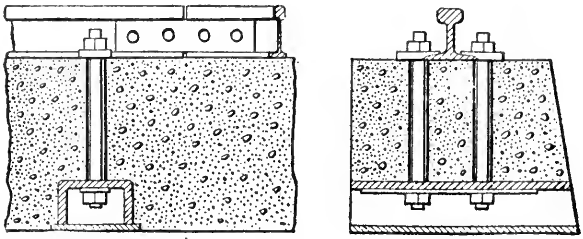


FIG 113.—ABANDONED METHOD OF FASTENING RAILS.

tracks for the amygdaloid mines is standard practice, unless the footwall gives trouble from swelling. These were first used by the Ahmeek, and the method of fastening the rails adopted from that mine by the Mohawk is shown in Fig. 113. The method has two disadvantages. In the first place a large amount of metal is used, since at 3-ft. intervals there is a channel through the stringer to give access to the track bolts, and these bolts themselves are carried in pipes and require bottom plates

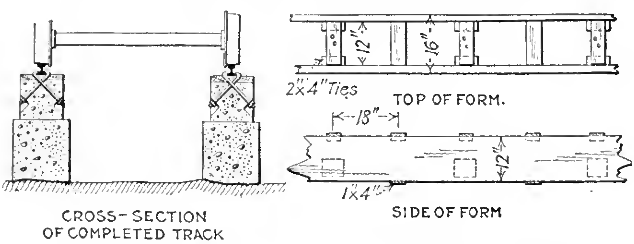


FIG. 114.—IMPROVED TYPE OF STRINGER AND FORM.

and beveled clips to grip the rails. A more serious difficulty is the fact that when a skip jumps the track it is liable to cut the heads of the bolts, which cannot then be extracted and replaced except by cutting the bottom nut or the bottom of the bolt itself.

To remedy these difficulties, the form of stringer shown in Fig. 114 was devised by W. F. Hartman, engineer for the Mohawk and Wolverine companies. Its advantages are manifest. The use of channels and clips is eliminated, the bolts are accessible and, in the most improved form, the

use of pipes to carry the bolts is avoided by inserting ⅞-in. bolts in the forms and withdrawing them when the concrete is partly set, leaving a suitable hole for inserting the ¾-in. bolts. The head of the bolt is also less exposed. The bolt holes are enough larger than the bolts so that an adjustment of ½ in. in the alignment of the track is possible. The system also involves the use of 2 × 4-in. creosoted ties at 18-in. intervals to take the shock of the skip off the concrete.

The arrangement of the forms for the 12 × 12-in. stringer is shown in Fig. 114, and the method of supporting these and the forms for the foundations of the stringers, is shown in Fig. 115. The 2 × 4-in. ties bind the tops of the form sides during pouring and 1-in. pieces across the bottom perform a similar function there. The recesses for the bolts are formed by small blocks of wood nailed to the sides at the proper intervals. The forms are supported on 2 × 4-in. pieces extending across the shaft. Vertical 2-in. planks, nailed to the form sides and extending to the shaft bottom, serve as forms for the concrete foundation. Before concreting, the shaft foot-wall is carefully cleaned. The forms are set in place by timbermen and lined up by a surveyor for direction and dip.

The concrete is mixed on the level above and slid down in 18- to 24-in. troughs, to a shoveling platform at the point where used. The work is done in 100-ft. lifts. A crew of from six to eight men can pour 100 ft. of double track in 24 hr. As fast as the form is filled, the cover is nailed on. The forms are left 36 hr. or longer. The concrete is mixed 1:2:5; poor rock from the mine is used, crushed in rolls to pass 2 in. and not screened or cleaned.

Concrete Stringers in Steep Inclines.—The installation of a concrete skiproad in a steep incline is a much more serious problem than a similar installation in a flat incline. The concrete stringer itself must be firmly anchored to the foot-wall and there is a constant tendency for the rails to creep down the shaft.

In the Copper Range shafts, which dip at an angle of 70°, concrete stringers of the Ahmeek type were tried and found unsuitable. The hammer of the skips is increased by operating at a steep angle, since they swing more on the rope and ride less smoothly. As a result, the nuts loosened on the bolts, which held the rail to the concrete and an elliptical hole was soon worn by the bolt in the concrete. Inspection of a 70° shaft is difficult and it was necessary to devise some method of holding the rails, which could be depended on for at least a week at a time, until the Sunday inspection took place.

The system illustrated in Fig. 116 was originated. Instead of being laid directly on the concrete or on wooden crossties, the rail is spiked to a continuous 6 × 10-in. wooden stringer lying longitudinally in the concrete base and occupying the upper, inner corner of it. The rails are notched

in three places on each side of a 30-ft. length to receive the spikes, which are thus better able to resist motion down the shaft. A cast-iron brace is spiked against the outside of each rail end to prevent spreading, as shown at *B*. The wood stringers are bolted to the concrete at about 8-ft. intervals. Access to the bottoms of the bolts is had through 3 × 4-in. galvanized-iron boxes, which are set in the concrete, but do not extend through it. The bolt is carried through a tube and is countersunk at the upper end, the countersunk hole being covered by the base of the rail so that the bolt tops cannot be broken off.

The concrete is held to the foot-wall by anchorages, as shown at *A*. These are put in about every 25 ft. They consist each of two pairs of

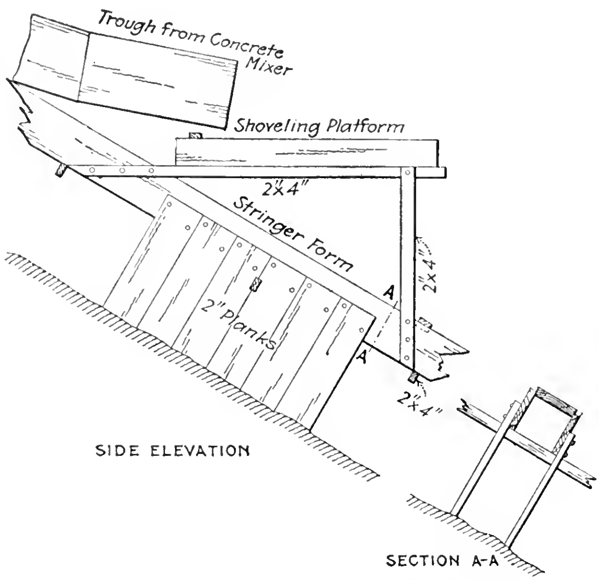


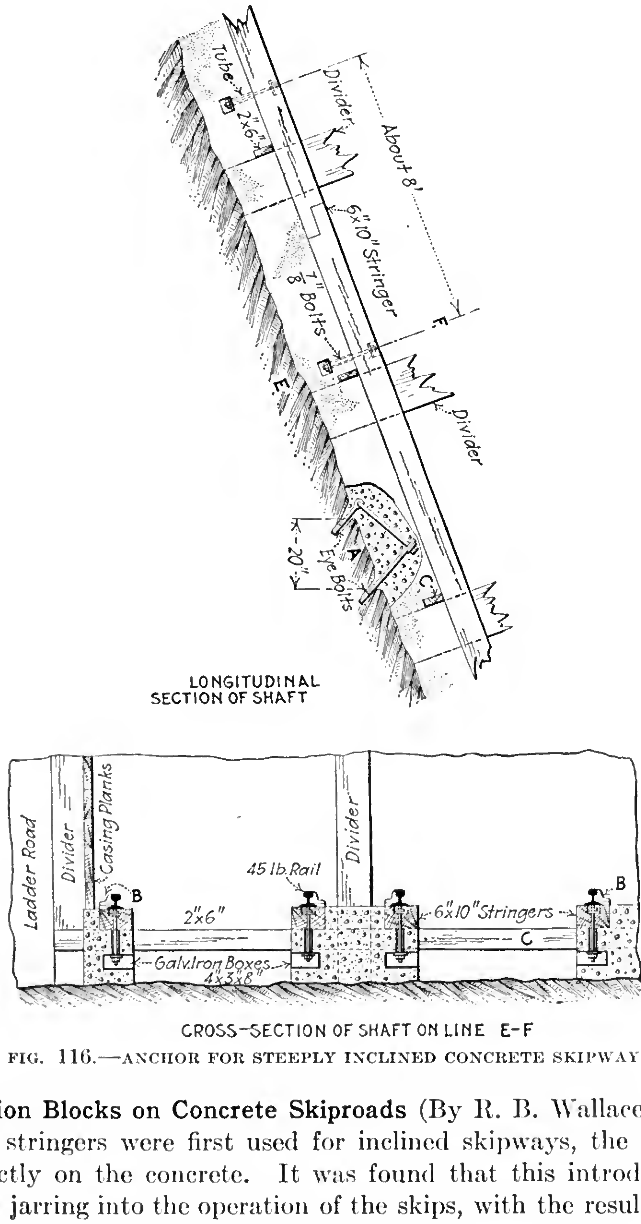
FIG. 115.—SUPPORTING STRINGER FORMS IN SHAFT.

eye-pins set in 10-in. holes drilled in the foot-wall and held by the concrete cast around them. The lower pins are fastened to the upper by bolts and to increase the bond, four short pieces of drill steel run across.

The concrete is cast flush with the top and side of the wood stringers, is continued to the wall or the divider in the case of the outside stringers and poured as one block for the adjacent stringers of the two skipways. This is in order to leave as little opportunity as possible for a rock to lodge and derail the skip.

In constructing the skiproads, the foot-wall is cleaned and 2 × 6-in. crosspieces *C* fastened to the dividers, are set to grade by the surveyor. The wood stringers are laid in the proper position on these, the bolts and

galvanized-iron boxes set in, and the forms built up of 2-in. planks. The 1:2:6 concrete is mixed at the bottom station of a 200-ft. lift and hoisted in a bucket to the shoveling plat from which it is transferred to the forms. This interferes less with mining operations.



CROSS-SECTION OF SHAFT ON LINE E-F

FIG. 116.—ANCHOR FOR STEEPLY INCLINED CONCRETE SKIPWAY.

Cushion Blocks on Concrete Skiproads (By R. B. Wallace).—When concrete stringers were first used for inclined skipways, the rails were laid directly on the concrete. It was found that this introduced considerable jarring into the operation of the skips, with the result that the

skip repair bill began to mount. To remedy this, it was found necessary to introduce a cushion between the rails and the concrete. A convenient method of doing this is illustrated in Fig. 117. In 1 is shown the position of the skip and the stringers in the shaft; in 2 and 3, the details of the arrangement. Wooden crosspieces *A*, 2 × 4 in. or larger, are laid in the concrete, projecting slightly above it, and the rails bolted to them. An opening *B* through the stringer directly below gives access to the bolts so that the blocks are readily renewable. If the concrete is already laid, the desired cushioning can be had by introducing ¾-in. pieces of wood under the rails, where they are bolted to the concrete.

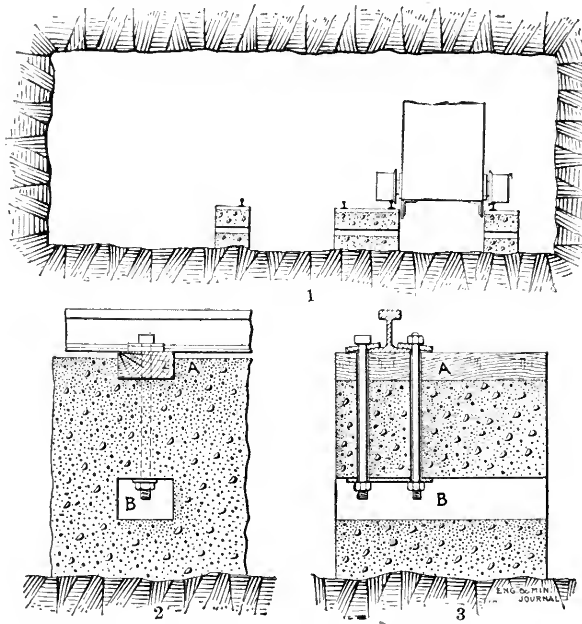


FIG. 117.—WOODEN TIES IN CONCRETE STRINGERS.

STATIONS

Cutting Station and Pocket in Ore (By L. D. Davenport).—In preparing for the cutting of a station and pocket in an orebody on the Mesabi range, lines for two small drifts are set on the wall plates 1 ft. inside of the studdles. The procedure is shown in Fig. 118. Two 5 × 7-ft. drifts are started on these lines, and at the same time, a small raise is started at a distance of 20 ft. below these drifts. A temporary chute is built at the bottom to direct the ore into the bucket in the shaft. This raise runs up along the shaft, and as the blocking and wedges are encountered, they are removed, and the lagging is nailed in place against the wall plates.

The drifts run in 16 ft. from the shaft, and when the raise is up on a level with their bottom, holes are knocked through. This raise is used as a chute while the station and pocket are being cut. When the drifts are in 16 ft. from the outside of the shaft, sills are laid perpendicular to the wall plates with the ends butting against the studdles. The tops of the sills are made flush with the top of the wall plate and the sills wedged tight against the shaft. The backs of the drifts are taken up to about 15 ft. to allow caps to be put in place. There are four 18-in. posts 11 ft. long put under each cap. The caps are 2 to 2½ ft. in diameter, and 10 ft. between the shoulders. The shoulders are cut in about 2 in.

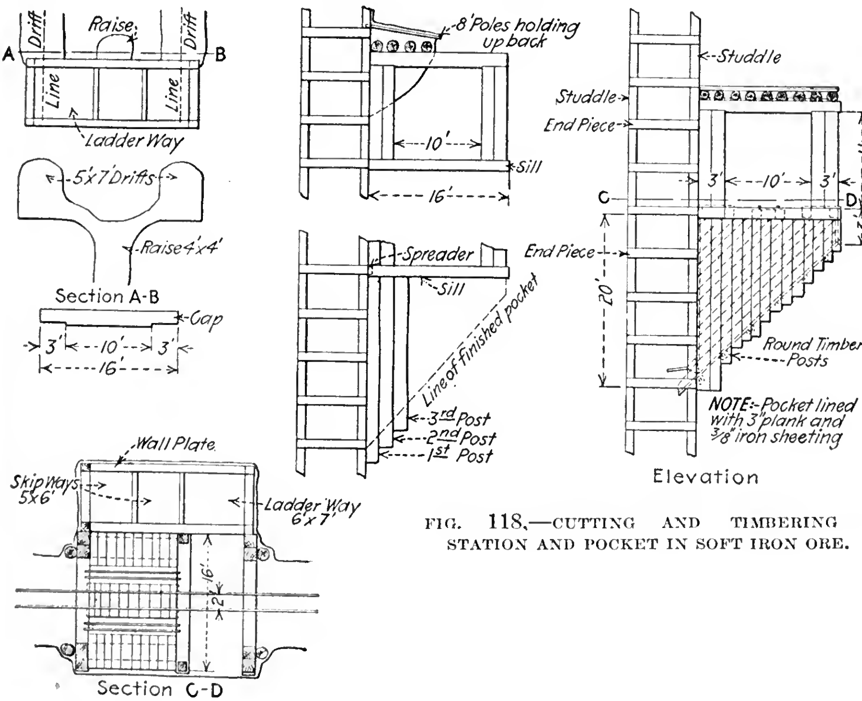


FIG. 118.—CUTTING AND TIMBERING A STATION AND POCKET IN SOFT IRON ORE.

The posts next to the shaft are placed against the studdles, and those at the further end are wedged in place. Each cap is also wedged against the shaft. To put the 20-ft. lagging in place (12-in. round timber), a cut is made about 2 ft. higher than the caps, and run in about 8 ft. from the shaft. Small poles are placed parallel to the caps, with one end resting on the wall plate and the other in a hitch cut for that purpose. These poles catch up the back while the lagging is being put in place. The ore in the upper part of the station is worked off and a small drift is started to allow the 20-ft. lagging to be placed. Two thicknesses of boards are placed over the lagging, with the joints broken, and the back is blocked

up above the boards. The remainder of the upper part of the station is taken out, and the rest of the 20-ft. lagging is placed and boarded up. Poles are used as before to catch up the back, one end resting on the lagging already in place and the other in a hitch. This 20-ft. lagging, 8 or 10 pieces, is equally spaced and small blocks are used to keep the pieces apart. The lower part of the station is then taken out, and a third sill is placed between the others at the ladderway divider. A "taking-up set" is placed on this sill. The sills are 2-ft. timber hewed flat on two sides, and the taking-up set is 14 × 14-in. square timber.

The pocket is then cut down next to the shaft, and posts are set under the sills to stand against the studdles. These posts are set in hitches about 2 ft. below where the bottom of the finished pocket will come. The pocket is next cut back, and other posts are placed against the first, under the sills. These posts are wedged tight into place and a spreader or cross-sill 6 × 12 in. is placed between the 16-ft. sills and against the wall plate to keep the 16-ft. sills apart. Another spreader 12 × 12 in. is placed at the other end of the 16-ft. sills and four more cross-sills are placed between. Two of these support the rails and are let into the 16-ft. sills. The four cross-sills are 16 × 16-in. square timber. The two wall plates in front of the station are cut off against the studdles and raised up even with the caps. The pocket is lined with 3-in. plank and 3/8-in. sheet iron. Quarter-pans are placed in the skipways.

Underground Crushing and Loading Arrangements (By Albert E. Hall).—In order to facilitate hoisting and avert delays in tramming at the Creighton mine, Ontario, a change was made from the system of dumping from the cars into the skips to that of dumping into a pocket. As the muck at Creighton comes through the chutes in large pieces, a pocket previously tried proved unsuccessful, since after drawing the muck from the chutes it would block in the skip-loading chutes. For this reason, and also because the rockhouse crushers could not handle the large material, making blasting and long stops necessary in the rockhouse, an underground crusher was installed. A separate pocket is provided for rock, which is smaller than the ore pocket. When the rock pocket is filled, the regular hoisting of ore is interrupted and the pocket emptied. The only change necessary is the opening of the rock dump in the headframe, which is done in a few minutes by the surface skiptender when the signal for hoisting rock is rung. Tramming of ore goes on just the same while rock is being sent up.

All ore is hoisted from the large pocket on the lowest level. The shaft has four compartments, but only two of these are used for hoisting ore, and the two skips work in balance. To get the ore from the levels above, a large raise, known as the ore pass, was put up. At the bottom of this is a heavily constructed chute with a gate consisting of inclined

rails on each post and a cross-timber. The chute feeds to a 10-in. grizzly, which in turn delivers to a chute at right angles to itself. This chute feeds the crusher, and on the side opposite the chute grizzly there is another grizzly, also 10-in., to take the muck from the cars on the level. A plan of the arrangements and a vertical section are shown in Fig. 119. The construction is heavy, but the ore to be handled is very heavy, being

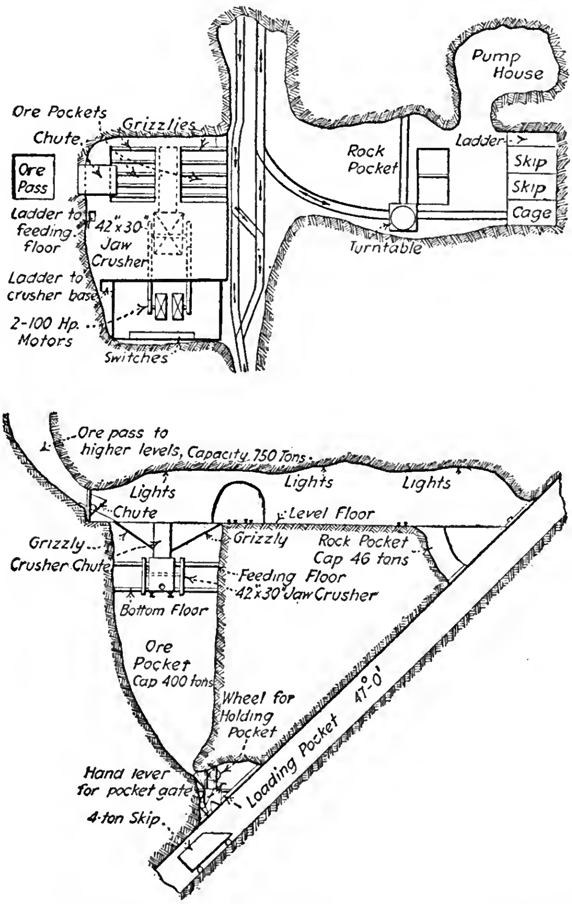


FIG. 119.—ARRANGEMENT OF UNDERGROUND STATION, CRUSHER AND POCKET.

for the most part pyrrhotite mixed with a little chalcopyrite and a small quantity of pentlandite. In addition to this, it is very coarse. The grizzlies and crusher chute have little inclination, but the material strikes them with considerable velocity and retains enough momentum to carry it to the crusher. Were they any steeper, the crusher would be blocked frequently.

The pocket holds about 400 tons and the ore pass will hold about

750 tons, so together they have a considerable storage capacity. The pocket is provided with a hand-operated gate of the arc type, with the convex side toward the muck. It moves up to allow the flow of muck into the loading pocket. The skips hold 4 tons and so do the loading pockets. The gate of the loading pocket is operated from the same floor as the gate of the main pocket by means of a pilot wheel and chain and sprocket. The front of the skip is made lower than the other three sides to facilitate even loading, and a depression in the rail serves the double purpose of acting as a skip chair and of giving the skip a steeper inclination, thus making more complete loading possible. A block with rail to fit can be swung into the chair when it is desired to take the skip to the levels being developed below. The loading pocket is built

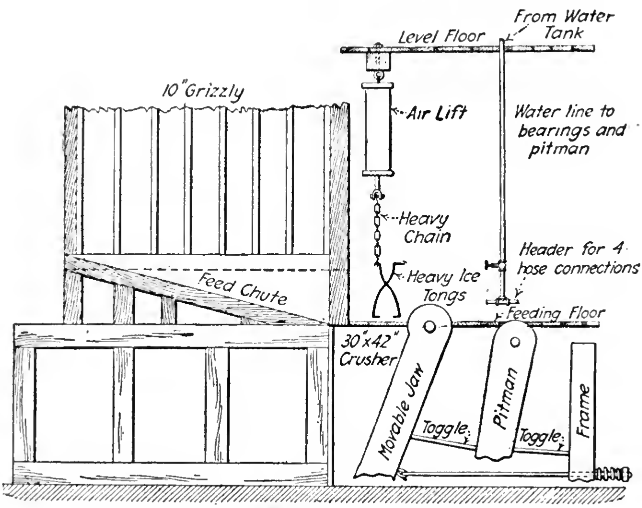


FIG. 120.—ARRANGEMENT OF CRUSHER AND BIN.

of boiler plate and was constructed on the surface, taken down in one piece and put into place.

The arrangement of the crusher is seen in Fig. 120. It is a Farrel crusher of the jaw type. Two heavy 18-in. I-beams run across the pocket and are cemented into hitches, giving a firm foundation for the crusher. All the timbering in the pocket to support the grizzlies and crusher chute is 14 × 14 in., and is protected from wear by boiler plate. The crusher is 42 × 30 in., and is set to deliver 10-in. material. Two 100-hp. motors are needed to run it. No oil is used in the operation of the crusher, a water pipe having been provided with connection to each bearing and to the pitman. Grease cups are used in addition.

Once in a while a large boulder will get wedged on top of the crusher jaws, and then an air-lift provided with a chain and a pair of ice tongs,

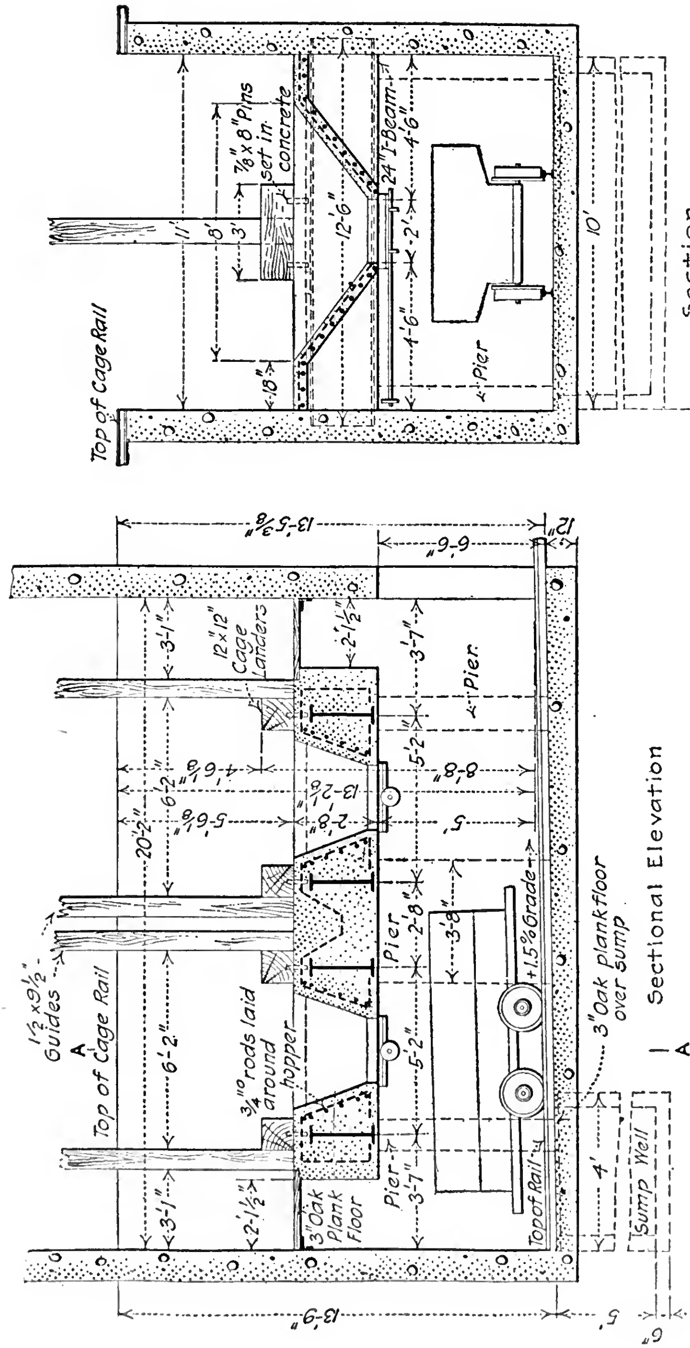


FIG. 121.—LONGITUDINAL AND TRANSVERSE SECTIONS OF SUMP UNDER MAIN SHAFT.

as shown, proves a great time saver. A large piece can be swung to fit the opening in a short time, whereas it would be impossible to bar it through and blasting means a great loss of time, since in addition to the preparation of the powder, etc., all the electric lights must be taken out, and even then the wires may be injured. If the piece is very large, enough powder to break it cannot be used without danger of injuring the crusher. The capacity of the crusher has never really been tested, as it is much greater than that of the hoist.

Débris Hoppers under Hoisting Compartments (*Coal Age*).—The main shaft of the Bunsen Coal Co., of Danville, Ill., contains two hoisting compartments. The shaft is vertical, of rectangular section, 20 ft. 2 in. by 11 ft., 204 ft. from collar to bottom level, with a 21-ft. sump below this. The hoisting compartments terminate below in a horizontal concrete and steel partition, constructed as shown in Fig. 121. This is in the form of a double hopper and below it sufficient room is left to permit a standard car to be run in. A heavy steel sliding gate at the bottom of each hopper is operated by means of a chain and wheel at the side of the sump. By this means the accumulated droppings from hoisting operations can be run out as desired into a car. When a car is filled it is pulled by motor through a concrete-lined tunnel on a sharp incline up to the haulage road. The walls and floor of the sump are of concrete 12 in. thick. At the low end of the sump bottom, extending across the full width of the shaft, is a well 4 ft. wide which catches all seepage water and provides ample room for a pump-suction line.

Spillage and Sinking Pocket (By Albert E. Hall).—Where skips are being loaded, either direct from cars or from a pocket, there is always some spillage. When work is going on below the loading point, it is necessary to keep this spilled material from falling on those below, and a pentice or bulkhead is often used for this purpose. If, however, the skips must make trips to the lower workings, the labor of removing and replacing a pentice causes loss of time. Even where the skips do not make trips lower than the loading point, the material spilled in loading has to be cleaned up at intervals, which is slow and costly work.

The pocket shown in Fig. 122 serves the double purpose of catching the spillage from the loading pocket and also of acting as a pocket for the material hoisted from the shaft bottom during the sinking. The shaft in question is inclined at 47° and has four compartments, a ladderway, two skip compartments and a cage compartment. The cage runs to the sixth level, so that the cage compartment is clear below this point. The skips run to the seventh level, now the lowest level; below this point the shaft is being sunk. The loading station lies between the sixth and seventh levels. Fig. 122 shows the general arrangement.

A drift was run in behind the shaft on the foot-wall side and a pocket

raised to the shaft just below the loading pocket. This was divided into three parts and chutes were built in the drift to correspond to these divisions. The center chute catches the ore spilled while loading the skips above. This is drawn from the chute and trammed around to the pocket on the seventh level. The other two chutes in the drift handle the rock from sinking. Two air hoists are used to hoist this rock to the pocket. Just above the spillage pocket, sheaves were set, one in the cage compartment and the other in the ladderway. The setting of the last was made possible by moving the ladder as near as possible to the wall. The rock drawn through the chutes is trammed around to the pocket on the seventh level. No extra labor is necessary with this arrangement. One hoistman handles both hoists and the buckets dump automatically.

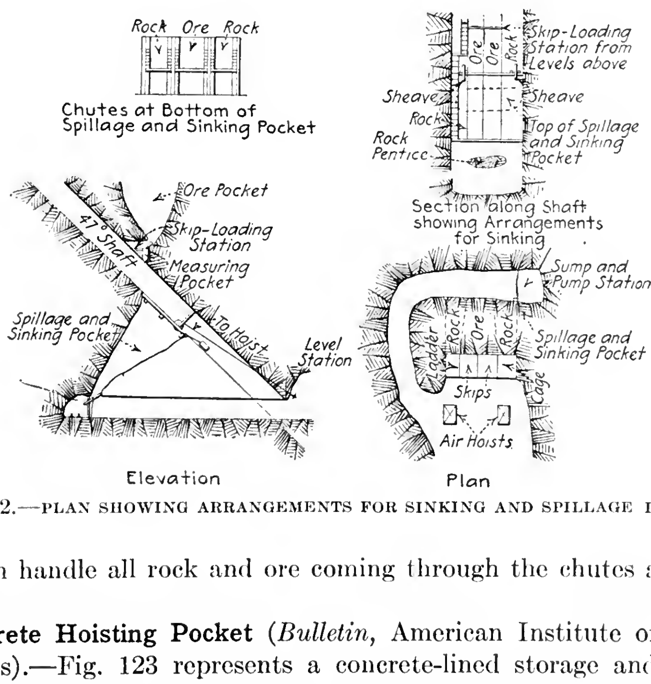


FIG. 122.—PLAN SHOWING ARRANGEMENTS FOR SINKING AND SPILLAGE POCKETS.

Two men handle all rock and ore coming through the chutes and tram it out.

Concrete Hoisting Pocket (*Bulletin*, American Institute of Mining Engineers).—Fig. 123 represents a concrete-lined storage and loading pocket of the Sacramento shaft, the principal hoisting shaft of the Copper Queen company. It is believed that the elimination of upkeep cost on the concrete lining will more than compensate for its increased first cost over a wood lining. Furthermore, the shape of the pocket and the smooth surface make the lining particularly adaptable for handling the wet sticky aluminous ores, for which purpose it was installed. The pocket starts with an elliptical bell-mouth and tapers to a 3½-ft. cylindrical chimney. This enlarges to a circular pocket 14 ft. in diameter and about 30 ft. high. The capacity of this, the pocket proper, is 52,000 cu. ft. or

260 tons. It has a conical bottom with a 6-ft. opening which feeds to a small pocket, which in turn supplies the measuring pocket. From one of the shaft compartments a drift runs around to a peep-hole into the pocket where the cylindrical chute widens to the pocket proper. The slanting bottoms at various points in the pocket are covered with rails where they are subject to the impact of falling ore. The concrete lining is reinforced in some places.

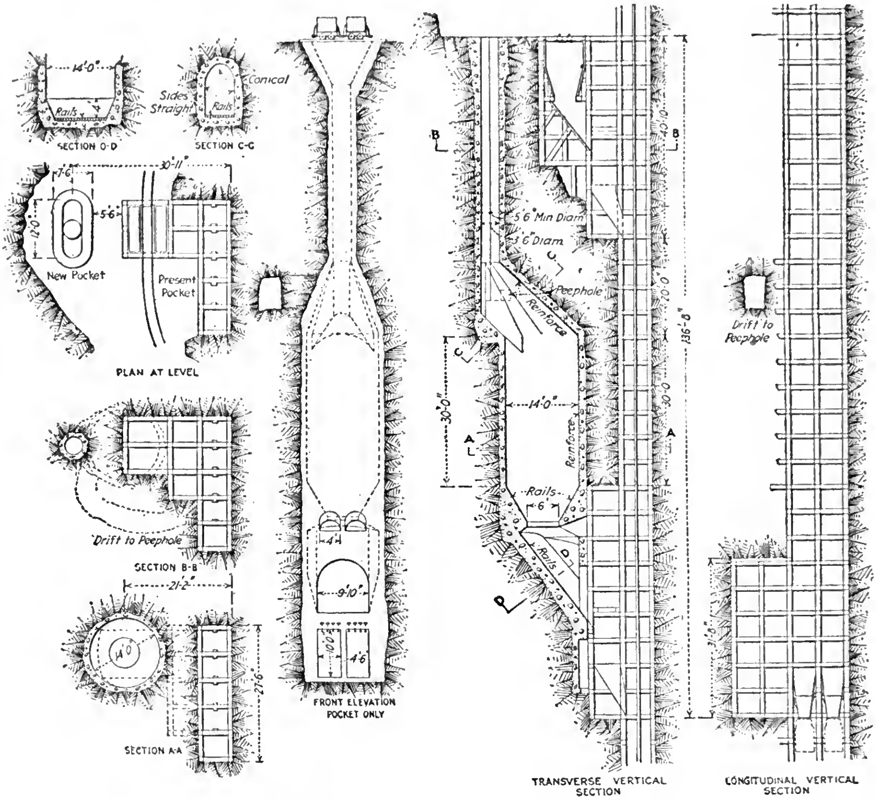


FIG. 123.—SECTION OF NEW CONCRETE-LINED SKIP POCKET AND OLD TIMBERED POCKET.

Concrete Shaft Station, Wolverine Mine (By Claude T. Rice).—At the 21st level of the Wolverine No. 4 shaft, which is sunk in the foot-wall, leaving a brow of rock between the station and the lode, the ground became heavy over the station and caused much trouble. It was necessary to retimber the shaft frequently, as timbers did not last longer than 6 or 8 years, failing by decay alone in that time. Whenever it was necessary to retimber, it was also necessary to remove much of the ground, which caused a repeated enlarging of the arch over the station. In an attempt to overcome this difficulty, reinforced concrete was adopted

and the station which was put in has now stood for over a year, with no sign of failure in the concrete.

The station is illustrated in Fig. 124. It will be observed that there are two heavy tongues of ground, one between the crosscut and the vein and the other between the floor of the crosscut and the roof of the shaft. Before the upper one could be supported by concrete in the crosscut, it was necessary to place reinforced pillars under the lower one. Unless

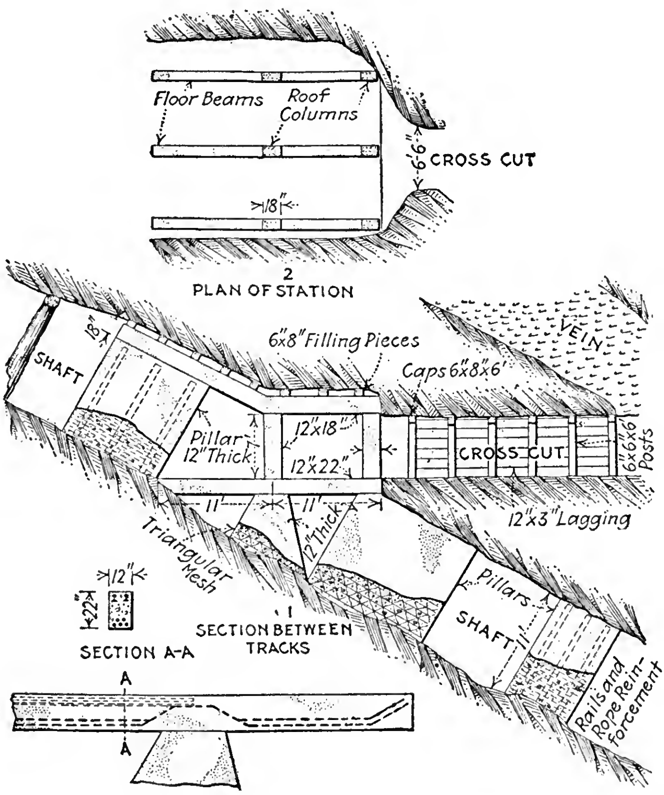


FIG. 124.—CONCRETE SUPPORTS FOR A STATION IN HEAVY GROUND.

these two tongues were firmly held in place, there would be a tendency for the whole piece of ground to slide down the shaft.

For these concrete pillars a mixture of about 1:2:6 or 7 was used, reinforced by such suitable iron scrap as lay about the mine, old rails being the most available. There are, however, two objections to the use of scrap for reinforcement. It should not be rusty or the bond between it and the concrete will be defective; and furthermore, it is necessary to watch the workmen closely to insure that the reinforcement will

be put in properly and in the desired quantity. Strands of old hoisting cable serve admirably.

Another method of reinforcing pillars is to use made-up reinforcement forms, which can be built into columns at the surface or nailed to the forms before the concrete is poured. Triangular-mesh reinforcement is handy, owing to the fact that it is hinged at each of its main members. Expanded metal is also good, but Hy-Rib, although it saves the cost of forms, is too expensive to use underground. Mr. Hartman, the company engineer, prefers triangular-mesh reinforcement; he uses style 42, which is 4-in. mesh material, folding nicely into reinforcing columns for the sizes of posts and pillars generally required in mine work. He has evolved standard forms that can be made up by the blacksmiths during spare time. For instance, triangular-mesh of the style mentioned, is purchased in the so-called 30-in. rolls, and can easily be made into columns 2-mesh, or 8 in. wide, by 5-mesh, or 20 in. deep, by wiring together two lengths of the reinforcement. Such a reinforcing column as this is stiffened by means of wire lacing running both crosswise and diagonally through it. It is possible for two men to make up ten 8-ft. columns of this kind in 2 hr.

In 1 is seen the triangular-mesh reinforcement. This is put in with its narrower side down, and is centered in the form. The concrete, which has been mixed usually on the level above and slid down the shaft in troughs to the platform, is then shoveled into the form, the sides of the latter built up and more reinforcement columns put in as the concrete rises. To guard against a plane of weakness developing in the finished pillar, the reinforcing columns are wired together. The work of shoveling the concrete into the forms and of tamping, is made easier by placing the reinforcement in the pillars crosswise instead of on end. If desired, the concrete pillars can be reinforced with rails and 3/8-in. strands of 1 1/8-in. discarded hoisting cable. In this case the old mine rails are placed at right angles to the plane of the shaft at intervals of about 3 ft. and the 3/8-in. strands of rope thrown into the forms in sets of three not over 12 in. apart vertically. The rails must not touch the ground or they are likely to take more than their share of the weight and crack the concrete. Before any of these pillars are put in, the loose rock is picked off the foot-wall and the ground washed down so as to insure a good tight bond between the concrete and the solid rock of the foot-wall.

The skip tracks at the Wolverine mine are close together, the manway being carried along the side of the shaft instead of in the center as is the practice in some of the Lake Superior incline shafts. For this reason, there is room for pillars only 12 in. wide, as they must be put in without interfering with the hoisting. On the side next the manway, the pillar is made somewhat wider.

Resting on these pillars and the foot-wall, the floor beams of the station are put in. To insure that they will not be injured by the skip's catching on them, they are made 12 × 22 in. in cross-section. The general manner of reinforcing these beams is shown in 3. The upper part is reinforced by two mine rails and five pieces of old 1⅛-in. hoisting cable are used in the bottom. Three of the ropes are turned up to an angle of 45° near their ends, and when passing over a support, in order to help the concrete in resisting shearing stresses. To increase the bond between the concrete and the ropes, the latter are allowed to fray at the ends. One of these beams lies between the two skip compartments, one between the manway and the adjacent skip compartment, and the other next the wall of the outside skip compartment.

The columns that support the roof beams are set over the pillars in the shaft. They are 12 × 18 in. in cross-section and are reinforced with triangular-mesh reinforcement, style 42. The roof beams which rest on these columns are reinforced in a similar manner to the floor beams, shown in 3, except that the rails are omitted. Transverse bracing beams are placed between the roof beams at the posts. They are 12 × 12 in. in cross-section; and are reinforced by three 1⅛-in. ropes, placed 1 in. from the under surface of the beam, the two outside ropes being turned up at an angle of 45° at the posts and ends to resist shearing stresses.

This whole structure is poured in place and care is taken to secure a good bond between the different members. A mixture of 1:2:4 is used, the rock being trap from the mine. It is crushed in rolls to 2-in. size and used without screening. The reinforcement is placed in the beams and pillars so that 1 per cent. of the beam area will be steel. Where wire rope is used, it is assumed to have half the strength of a bar of the same diameter and proper allowance is made for this in the area computations of the reinforcement requirements.

The concrete filling pieces are made 6 × 6 in. in section and from 7 to 8 ft. long, reinforced with six ⅜-in. strands of old hoisting cable. These filling pieces, as well as the posts, caps and lagging slabs used in the crosscuts to the lode are made in forms on the surface. In all these a mixture of 1 :2 :4 is used, and they are permitted to set and season at least a month on the surface before being sent into the mine. The posts of the drift sets are made 6 × 6 in. in cross-section and are 6 ft. long. They are reinforced by four ½-in. bars, which are held in the proper position with regard to one another by wrapping with ¼-in. rodding. The caps are made 6 × 8 in. in cross-section and 6 ft. long, reinforced with three pieces of 1⅛-in. cable. The lagging slabs are 3 × 12 in. in cross-section and 4 ft. long. A strip of triangular-mesh reinforcement, two meshes wide, is used in reinforcing them.

The ground in which these concrete sets are used is heavy, due to the

cracking of the brow between the lode and the crosscuts, which, as originally driven, was considerably larger than at present. The weight, however, is a regular one, and not so trying as that encountered in swelling ground. The sets are not cracking to any extent, and the lagging is standing well, bending slightly but without showing even hairline cracks.

Wood, Steel and Concrete Station (By Claude T. Rice).—The Hancock No. 2 shaft was sunk vertically to a depth of 2600 ft., using buckets to

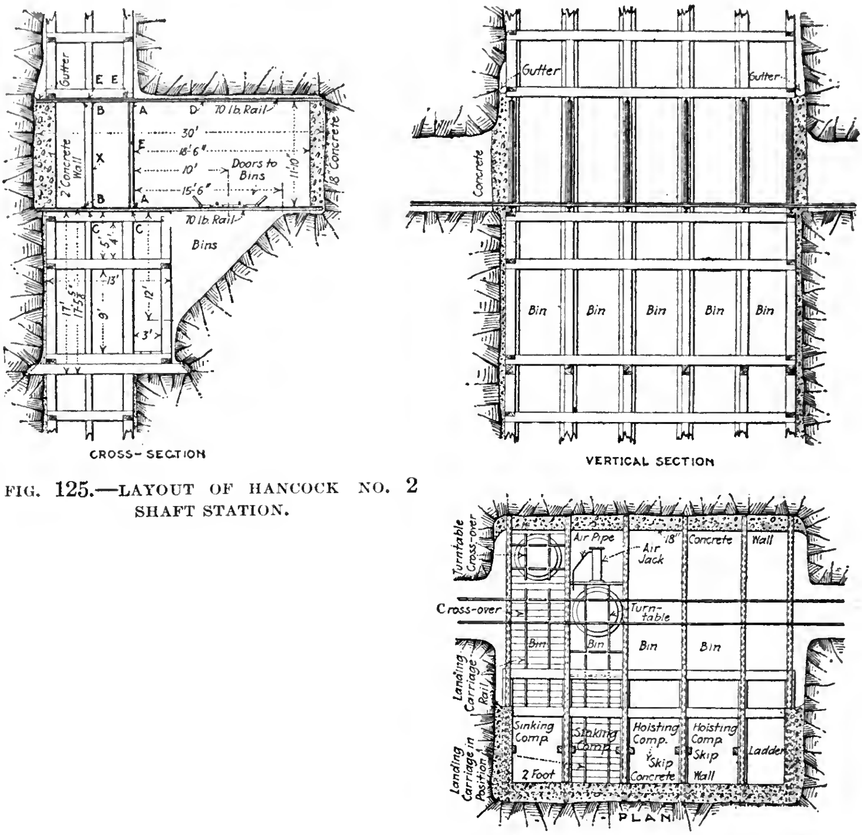


FIG. 125.—LAYOUT OF HANCOCK NO. 2 SHAFT STATION.

raise the broken ground to the surface. At about this depth the 34th-level station was cut, bins were built and thereafter sinking was continued, the muck or spoil being raised in buckets to the bins at the 34th level, from which it was afterward loaded into large skips and raised to the surface. The shaft was sunk with five compartments, four for hoisting and one for pipes and ladders.

Weathering of the rock was guarded against by concreting the sides of the station soon after it was cut. In supporting the roof of the station

no timber was used that would later have to be replaced, nor was concreting considered necessary. Instead, a series of 70-lb. rails, placed with their flanges downward so that they could be bolted easily to the supports, was carried across the station between the compartments of the shaft. The ends of these rest upon the concrete walls, while they are supported at the brow of the station by rails coming up from the floor rails. These sill rails, in turn, rest upon the timbers of the shaft at their middle and in hitches at the ends. These lower rails carry the floor of the station. The details of the station lining are shown in Fig. 125, in which the different supports are marked by letters.

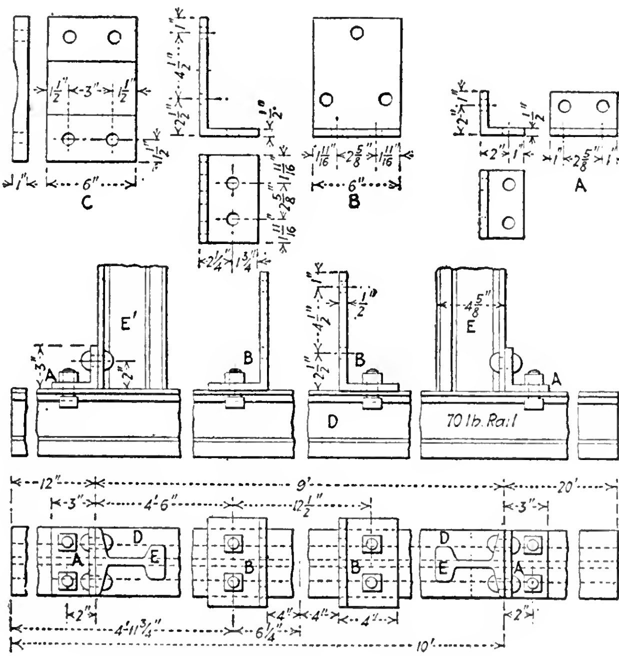


FIG. 126.—DETAILS OF STEEL SUPPORT FASTENINGS.

The details of the methods of fastening these rails together with the parts designated by the same letters are illustrated in Fig. 126. The long rails that are used as cap pieces to support the roof, as well as the sill rails that carry the floor of the station, are marked *D*. The sill rails are put in with the flanges upward, and the cap rails with the flanges down. The sill rails rest upon iron plates marked *C*, which are placed on top of the shaft timbers, so as to keep the rails from cutting into the wood, while on top of the cap rails similar plates, also marked *C*, are put in to carry the shaft timbers. The post rails marked *E* and *E′* in the case of the rail that is put in on the back side of the shaft when concrete is not considered

necessary, spring from these sill rails and are secured to them by the angle pieces A, which are bolted to the flange of each rail. At the top a similar angle fastens the cap rail to the posts. Also springing from the sill rails is a piece of 8 × 10-in. timber, the studdle to which the guides are fastened by lagscrews. This guide studdle is fastened to the sill by the plate B, which extends up higher on the timber than the plate A does on the rails. This plate is bolted to the rail and to the timber as shown in the illustration. In the flanges of the rails that are used as posts in the front of the station, bolt holes are drilled as shown in Fig. 126, so that the timbers that are to serve as jambs for the doors can be bolted to them, and to carry the partitions put in at each compartment to prevent anything from falling down the shaft. These partitions and doors are made of 2-in. lumber and do not require further description. Each door is equipped with a latch.

In cutting the station, it is the practice to cut out only for the stations at first, and then afterward, working in from the shaft, to break out the triangular space for the ore bins from below. The posts of the station-shaft sets are made 11 ft. 10 in. long, while posts 9 ft. long are put in at the bins. In timbering the shaft at the bin, a small set is put in in front of the shaft sets. These sets are 3 ft. wide in the clear, so as to provide room enough for the men to move about while loading the skips. A bearer set is put in just below the bin, and on these bearers are stood the timbers that carry the long posts in front of the skip chutes. The cross-timbers at the bins are carried in one piece clear across the shaft and over the front of the bin. Into these divider timbers, the cross-braces running in the direction of the wall plates, as well as the posts or studdles, are dapped.

This type of station and method of lining have become standard for the shaft, as it provides for everything with a minimum of excavation. The bins are fitted with doors, one on each side of the track, for the ore is hauled to the station in saddleback cars, by an electric locomotive. The doors on the side away from the station, where the drop is small, are opened first, so that the ore dumped on that side will slide down the bottom of the chute, made of rails laid in concrete, and form a cushion for the ore on the shaft side to strike upon, thus breaking its fall.

Concrete Station at Champion Mine.—The stations of the Champion mine sometimes require concreting of the back where rather large areas are exposed. Such a station is illustrated in Fig. 127 in plan and section. The wood timbers here shown are replaced in later stations by concrete beams or pillars. The peculiar shape shown in the plan is rendered necessary in order to provide for sinking, the hoist for that purpose being situated at A and a temporary turntable at B for a temporary track C. The permanent turntables E and the cradles D are placed as illustrated. The manway is covered with 10-in. timber shown at I; a trap-door F sets

in this and the 8-in. air pipe G passes through it. Between the skip compartments a brattice of planks H is built, to prevent miners from stepping off the cage into the empty compartment.

The reinforcement in the concrete roof of the main part of the station, as seen in the sections and in the plan of the roofs, consists of 1¼-in.

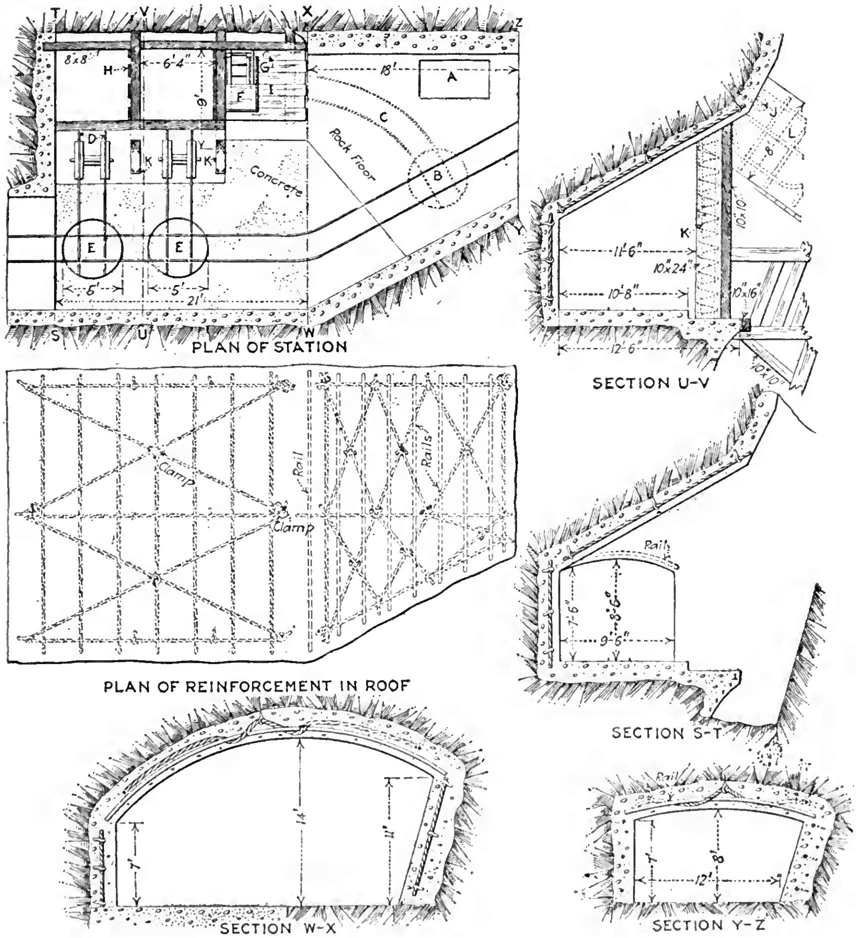


FIG. 127.—PLANS AND SECTIONS OF CHAMPION CONCRETE-LINED STATION.

rope. Some of this is threaded through eye-pins in the back, these being disposed as shown. This cable is drawn as tight as possible, and the ends and the crossings clamped. Above it are placed six pieces of the same cable extending in the direction of the shaft. The roof is about 18 in. thick. Similar reinforcement is used for the station sides, which are from 10 to 18 in. thick.

In section *UV* can be seen an eye-bolt *J*, in the brow over the shaft. One of these in each skipway is used to hold the blocks for unloading heavy timbers. The two supporting posts *K*, opposite the shaft dividers, are built of four 4-in. angles in the corners, wrapped continuously with ¾-in. wire rope, the wraps about 12 in. apart. These posts are 10 × 24 in. The divider in the shaft, seen at *L*, is of concrete, reinforced with 1¼-in. rope as shown.

The side extension of the station has an arched roof and includes eight 25-lb. rails in its reinforcement. These are bent to the arch of the station. Eye-pins are also set in the rock and a network of 1¼-in. cable laced through them over the rails, except that where it passes through the center pins, it is taken under the rails and up through the eye-pins again. The opposite side of the station also has a rail reinforcement in the back of the drift.

RAISING

Scaffolding in an Untimbered Raise (By Frank C. Rork).—A method of scaffolding applicable in driving an untimbered raise is shown in Fig. 128. The usual method is to cut a stull the right length and wedge it in; this is not an easy thing to do, working from a ladder, and more or less time is consumed in finding and cutting the timber, in measuring and in wedging the timber in place. By the method here illustrated, when the round is finished, four holes are drilled, two at *A* and two at *B*, near the corners of the raise. The depth of these holes should be about 10 in., depending on the nature of the rock. They should be placed the height of one cut above *C* and *D* and will then be in the proper position to hold the scaffold after the round is fired. The scaffold can be quickly erected on two stulls, resting on four pieces of steel inserted in the holes. As the raise progresses, the lower holes can be used to secure the ladders in the manner illustrated. The shallow holes can be drilled rapidly and easily with all the equipment at hand and the drill steel consumed would probably be lost or wasted if not thus utilized.

Five-hole-cut Raising Method (By H. H. Hodgkinson).—The five-hole-cut method here described has proved to be the best and most economical method for both drifting and raising at the New Jersey Zinc Co.'s mines at Franklin Furnace, N. J., where the ore and limestone as a rule are soft. Although this method necessitates drilling more holes than either the draw- or the V-cut method used in the West where the ground is hard and brittle, yet it can be depended upon to break a good clean heading in either ore or limestone. These other methods fail unless an excessive number of holes are drilled, which takes more powder, for due to the soft nature of the ground the holes when fired do not break the ground but simply chamber and blow off a little of the collars.

In drilling up a round in a raise, 17 holes are put in, as shown in Fig. 129, to a depth of 6 ft. The ground is easily drilled, the machines averaging 14 to 20 ft. per hour. When the round is fired a heading 5 × 5 ft. is broken with an advance of not less than 5 ft. This is accomplished in an eight-hour shift.

About 92 sticks of 1×8-in. 50 per cent. gelatin dynamite are required to break a round. Hole No. 2 is loaded with eight sticks of dynamite

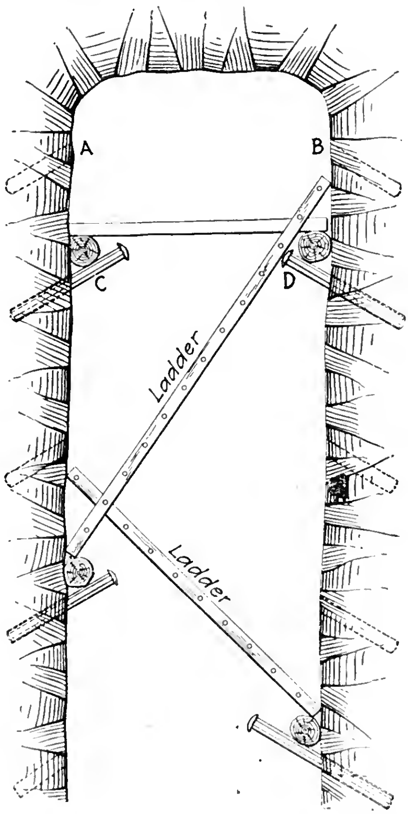


FIG. 128.—SCAFFOLD AND LADDERS SUPPORTED ON DRILL STEEL.

and fired; this breaks through the other four cut holes Nos. 1, 3, 4 and 5 as shown. Holes Nos. 6, 7, 8 and 9 are then loaded with seven sticks each and fired after the muck is cleaned out of the space made in firing hole No. 2. The remaining eight holes are loaded with seven sticks of dynamite each and fired. At least two bags of tamping are used in each hole and pressed in firmly.

Draining Watercourses in Chutes (By Edward P. Scallon).—In raising, watercourses are frequently encountered. If the raise is to be

used as a chute, much delay and annoyance in the subsequent handling of the product are occasioned by an admixture of the water and ore in the chute, and in most cases it is essential that this condition be remedied. At the Lincoln mine of the Inter-State Iron Co., at Virginia, Minn., the form of chute construction shown in the accompanying drawing, Fig. 130, is employed successfully to overcome the difficulty. After the raise is driven, several 6-ft. drill holes are placed in its side, so as to cut the water-bearing strata transversely at different elevations. Pipes are then ce-

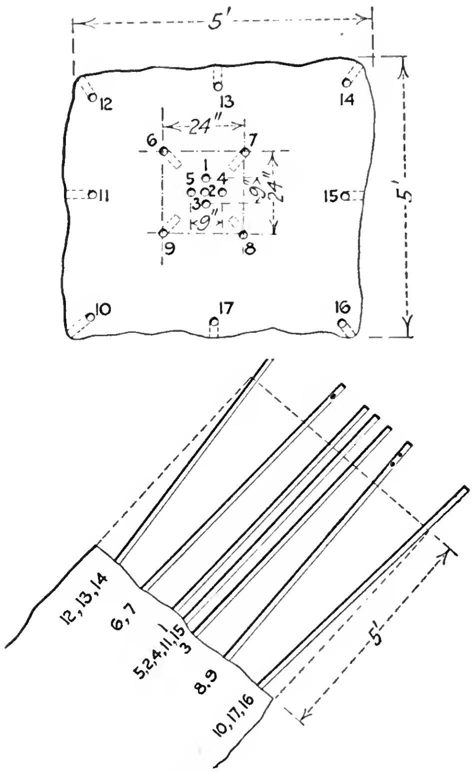


FIG. 129.—LAYOUT OF HOLES FOR FIVE-HOLE-CUT RAISING.

mented into the collars of these holes and extended, close to the side of the raise, to the level below. Air lines and any other necessary pipes are placed along the walls, and the entire raise is lined with concrete, corrugated culvert pipe being used as a form. All the pipes will of course be embedded in the concrete; the small drainage pipes from the drill holes will collect all the water and carry it to the level below. This prevents the exertion of any water pressure on the concrete while setting or subsequently, and also allows proceeding with the drainage of the area tributary to the raise.

By the use of these drainage pipes it is found at the Lincoln mine that almost all watercourses can be sealed off with oakum and dry cement. In this case, if desirable, a form of plank chute providing protection to the pipes and cement can be substituted for the more permanent concrete and metal-lined construction described above. The water collected

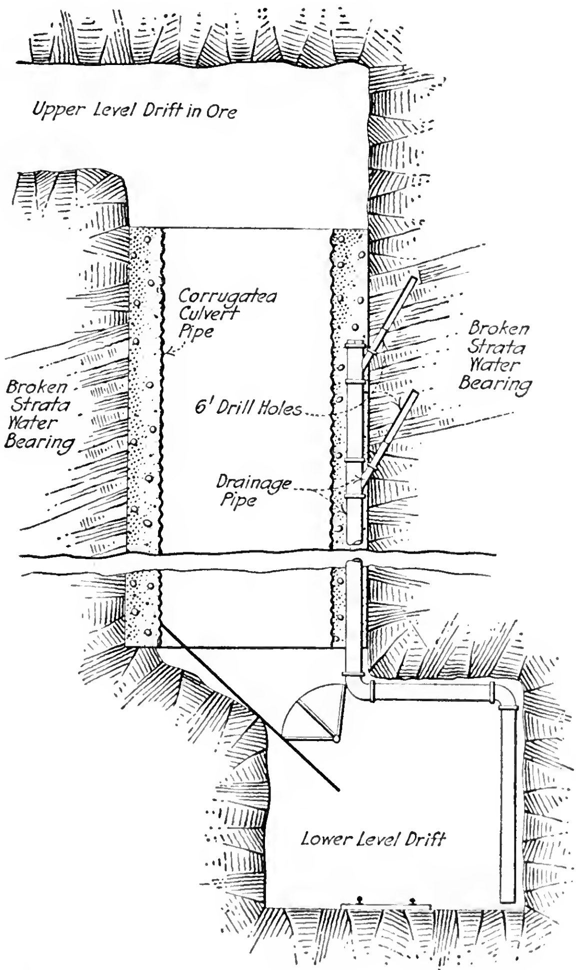


FIG. 130.—CONCRETE AND PIPES FOR DRAINING RAISE.

in the pipes from such a raise is used to supply tanks feeding Leyner drills on the lower levels.

LADDERS

Wood and Iron-pipe Ladder.—An economical and serviceable mine ladder may be made of 4 × 4-in. timbers, with rungs of 1-in. pipe, the

latter consisting of discarded pipe sawed into 20-in. lengths, Fig. 131. The rungs are spaced 1 ft. apart, holes being bored through the timbers

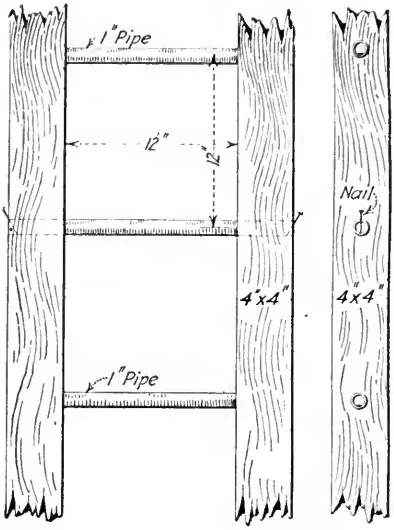


FIG. 131.—LADDER OF 4 × 4-IN. UPRIGHTS AND 1-IN. PIPE RUNGS.

just large enough to receive them. Small holes are drilled through each pipe length, one near each end, before driving into the timbers, and when

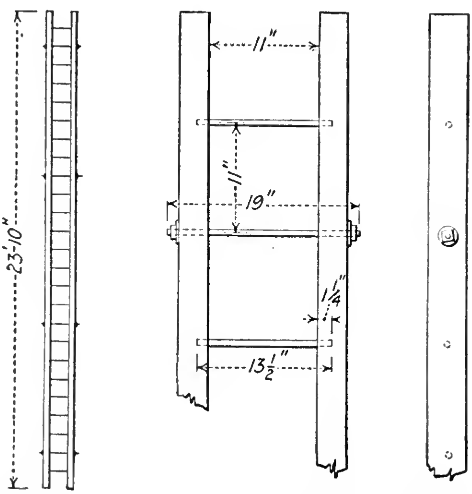


FIG. 132.—LADDER OF 3 × 4-IN. UPRIGHTS AND 1/2-IN. STEEL RUNGS.

the rungs are in place nails are inserted into these holes and driven tight. This prevents the rungs from twisting or slipping sideways and holds the ladder securely together. Such ladders cost more in the first place than

ladders of the same length constructed of 2 × 4- and 1 × 4-in. lumber, but their cost may be kept low by using the better parts of discarded pipe and having the shop crews make up extra ladders during what would otherwise be spare time, while their great durability, which largely reduces the repairs necessitated by breakage and wear, makes it economical to use them instead of all-wood ladders, especially in more permanent manways.

Steel and Wood Ladder (By Harold A. Linke).—In Fig. 132 is shown a substantial and inexpensive mine ladder, the sides of which are constructed of 3 × 4-in. stuff, surfaced two sides and one edge, and the rungs of ½-in. round mild steel. The bill of material is as follows:

Two pieces 3 × 4-in. by 24-ft. S2S1E.
Thirty feet ½-in. round mild steel, cut into 21 rungs 13½ in. long and four 19-in. lengths, the latter threaded 2 in. both ends.
Eight ½-in. cut washers.
Eight ½-in. square nuts.

After the ladder is made and the bolts tightened it is well to rivet the ends of the tie bolts or burr the exposed threads to prevent the nuts from working off.

Portable Steel Ladder (By L. O. Kellogg).—The Penn Iron Mining Co., on the Menominee range, has adopted an all-steel ladder for under-

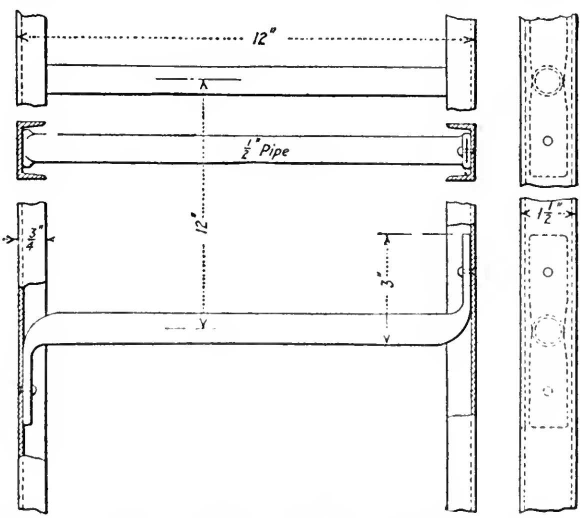


FIG. 133.—LIGHT ALL-STEEL LADDER OF PIPE AND CHANNEL.

ground use. Fig. 133 shows the type used in stopes and manways where the installation is more or less temporary. The ladder is 12 ft. long, 1 ft. wide and weighs a little under 50 lb. The sides are of 1¼ × ¾-in. channels with the flanges inside; the rungs are of ½-in. pipe. The ends of the pipes are bent at right angles in opposite directions, flattened and

riveted to the channels, the rivets being countersunk on the outside. The ladders are of unusually light construction, but are found to stand up well in service and are extremely convenient to carry about and place as needed. They are found preferable to the type first tried, built of heavier channels and 3/4-in. pipe and weighing nearly twice as much. For permanent installations, as in the shaft, a much heavier ladder is used, built also entirely of steel.

Pipe-and-angle Ladder (By Edward S. Wiard).—The details of the steel ladders used at the Capital mine, Georgetown, Colo., were worked out by the superintendent, E. C. Bauman. The principal advantages of these ladders as against wooden ones are lightness, portability, greater

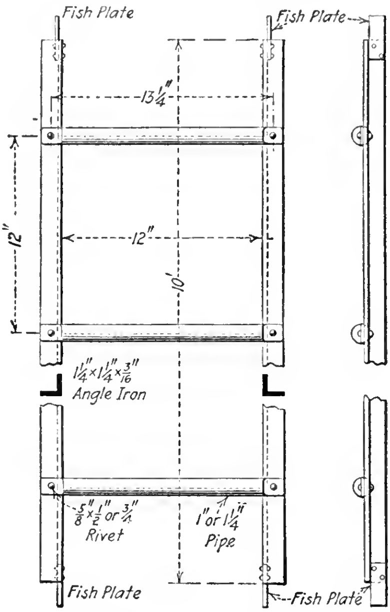


FIG. 134.—MINE LADDER OF ANGLE IRON AND PIPE.

cheapness in the long run and safety. Fig. 134 shows the construction and dimensions. The rounds are made of old 1-in. or 1 1/4-in. pipe. This is cut up into suitable lengths, and both ends, after heating in the forge, are mashed flat. The flattened ends are then drilled and cold-riveted to the angle irons forming the sides of the ladders. The ladders at the Capital mine are made in the 10-ft. lengths, which are easily handled. They are held in place in the manways by ladder hooks placed in the usual way. To hold the ladder away from the wall plates—if this procedure is necessary to give a good foothold—blocks may be inserted between the ladder and the wall, the hook embracing both block and ladder. If desired, four pieces of steel of the same size and shape may be

riveted in pairs to the ends of the ladder to attain the same purpose—that of holding it away from the sides of the manways. As fast as the ladders are put into place they are connected to the lengths above or below by fish-plates and bolts, so that each section receives the support not only of its own ladder hooks, but of all the others.

The weight and cost of this ladder at the mine are as follows:

	Weight in pounds	Cost
Rivets and bolts	3	$0.30
Angles	30	0.83
Pipe	27 (half price)	0.50
Fish-plates	1½	0.04
Total	61½	
Labor		0.86
Total cost		$2.53

If the pipe is considered worthless the cost will be reduced to $1.67. The material for wooden ladders will cost from one-fourth to one-half the steel; the labor cost will be as much or greater. The greater cost of the steel ladders is soon made up by the longer life.

Wood vs. Steel Mine Ladders (By George E. Collins).—In the *Engineering and Mining Journal* of May 29, was a note with sketch by my friend E. S. Wiard, about steel ladders used at the Capital mine, Georgetown, as to which I wish to suggest a word of warning. In most mines such ladders would be unsafe. Moisture is apt to collect behind the rungs where the latter are riveted to the sides. This results in rust, and when the rivet has rusted through the ladders are very dangerous, as the defect is not visible. Steel ladders should be used only where liability of dry rot renders wood unsuitable; and even then I believe a design in which the pipe rungs actually pass through the sides would be safer and more generally preferable. Even in a dry mine there is usually moisture enough to develop rust on metal surfaces, especially when two such surfaces are in approximate but not actual contact.

In my experience, light weight in ladders is ordinarily a matter of secondary importance. Where a length of ladder has to be moved frequently, very light ladders will of course be built. Ladders are intended primarily to travel on; and next to safety, convenience in climbing is the most important requisite. Pipe or other metal rungs are cold to the hands, and when wet and gritty cut the skin more than wooden rungs. Moreover, pipe is slippery and does not afford so good a grip for the feet as wood.

In most places, wooden ladders are best. Probably the most generally suitable design, taking convenience and cost into consideration, is that shown in Fig. 135, where sides and rungs are both made of 2 × 4 stock,

the rungs being inset 1 or 1½ in. into the sides. If neatly made of dry lumber, the rungs swell when wet, so that they are firmly held in the slots, even if the nails with which they are originally secured should rust out. Worn rungs are easily replaced—an important consideration, for the neglect of which there is no excuse—and even when a rung is worn through or broken, the pieces hold sufficiently for safe travel. Such ladders made of red spruce, at a high-altitude mine in Colorado, cost 10½ cts. per foot. complete as against Mr. Wiard's figure of 25 cts. per foot for the steel ladders. Their life, apart from damage by falling rocks—which in my experience is equally destructive to the 1- or 1½-in. pipe—is very long, sometimes exceeding that of the mine itself.

Where the ladders are exposed to the drip of acid water, I have used the form shown in Fig. 136. Here the rungs are made of 2 × 4 or 2 × 5 lumber, sawed on an angle lengthwise. so as to make pieces of the section

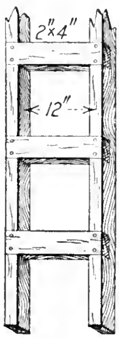


FIG. 135.

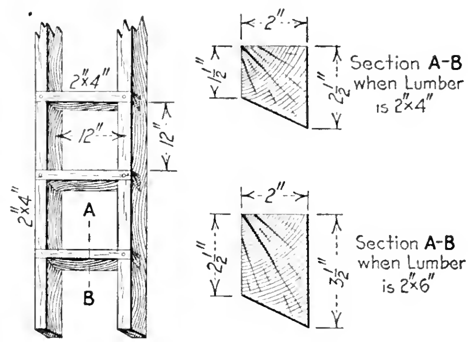


FIG. 136.

LIGHT WOODEN MINE LADDERS.

as illustrated 1½ in. one side and 2½ in. the other, when sawed from 2 × 4 lumber, or 2½ in. and 3½ in. respectively, when sawed from 2 × 6 lumber. In either case, one cut in the side pieces is made at right angles and the other obliquely, so that the rungs are firmly held without nails. Of course the flat side of the rungs is placed uppermost. These ladders cost but a trifle more than the first mentioned. Where there is liability of dry rot, I have used 2 × 4 lumber for the sides and old pipe sawed into lengths for rungs. If the holes in the sides are a tight fit, no other fastening is necessary. I do not know why wooden uprights last so much longer than rungs—to a greater extent than the increased section would suggest—but such, in my experience, is the fact. The sides might be dipped in creosote or treated with copper sulphate, although I personally have not tried either. Either preservative used on wooden rungs, would be a nuisance. The facility with which the sides of wooden ladders may be spiked to timbers or fastened to one another by means of cleats is

much greater than that by which steel ladders can be suitably fastened by hooks or staples. Of course the advantage mentioned by Mr. Wiard, of each section being supported by the others, may and usually does apply to wooden ladders as well as to those made of steel. In the old district of Gilpin County, ladders usually have sides of 2 × 4 lumber, with rungs made either of round 1¼-in. hardwood or of 2 × 2 native lumber chamfered at the edges and turned down at the ends to 1¾ in., which fit closely into a round hole of equal diameter in the side pieces. A nail through the

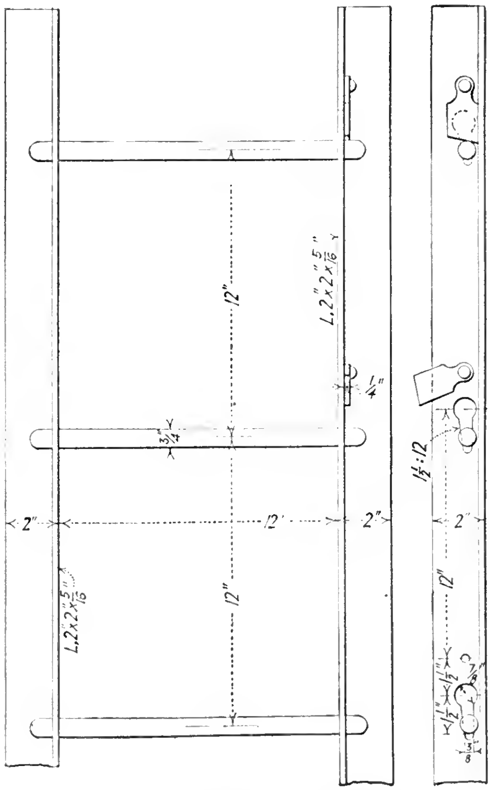


FIG. 137.—LADDER OF ANGLES WITH REMOVABLE RUNGS.

side into the rungs at each hole adds some further security. These are good ladders to climb, but are not quite so safe as those made of 2 × 4 slats nor so durable. They retail at the rate of 12 cts. per running foot. Slats made of 1 × 3 or 1 × 4 are sometimes used, but are not strong enough, and give poor footing. They should be used for movable ladders only.

Isabella Knockdown Iron Ladder (By Leo H. P. Kneip).—The ladder shown in Fig. 137 is one installed in the new steel and concrete shaft of

the Isabella mine, at Palmer, Mich., operated by the Cascade Mining Co. The ladder was devised by Francis H. Tippett, as a result of a competition promoted by the superintendent, Thomas J. Nicholas. The ladder uprights are 2 × 2 × 5/16-in. angles, the rungs are 3/4-in. iron. The rungs are inserted through 7/8-in. holes in the angles and drop into a taper slot cut downward from the hole. A pair of grooves in each end hold the rung in the flange against turning and also brace the angles against side motion. The rung is locked in with a swing lug having a beveled bottom, as shown.

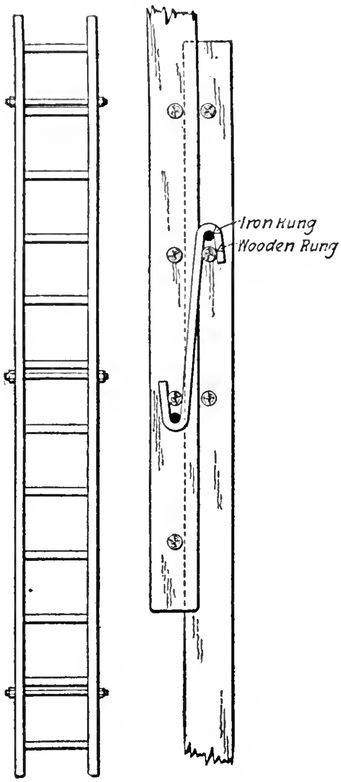


FIG. 138.—METHOD OF HANGING LADDERS IN A RAISE.

If a rung should break, a new one can be inserted without taking down the ladder or springing it. The ladder contains no threads, bolts or nuts and can be made without special tools in any mine shop.

Method of Hanging Ladders.—When ladders are required only for temporary use in raises and millholes in the mines of the Copper Range Company, in the Lake Superior country, it is the custom to use several 12-ft. ladders; the lower one rests upon the ground while each of the others is carried by 5/8-in. round-iron, S-shaped hooks from the ladder below, as

shown in Fig. 138. The ladders are made of oak with 2 × 4-in. legs and rungs 1½ in. in diameter spaced 1 ft. apart. Three tie-rods are used: One at the center; one below the second rung from the bottom; and one above the second rung from the top. The S hooks grip the ladders by the tie-rods and the ladders overlap at the junction. It is better to place the overlap of the lower ladder on top of the upper one, provided they are inclined, as they should be, since that arrangement makes it easier for a

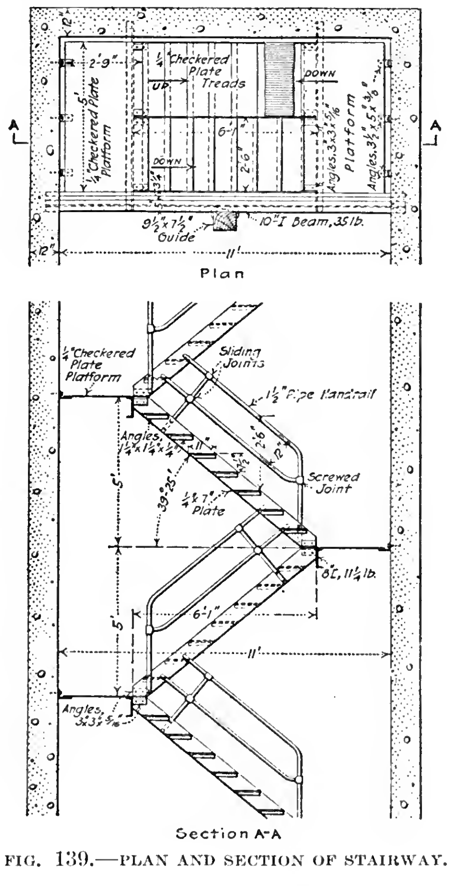


FIG. 139.—PLAN AND SECTION OF STAIRWAY.

man coming down to know when he has reached a junction. A temporary sollar is usually put in the raises at intervals of every two ladders whenever three or more ladders are used. Sprags are used to prevent side swing. The overlap is so long that there is no danger of hinging of the ladders at the junctions.

Vertical-shaft Steel Stairway (*Coal Age*).—The air shaft or auxiliary shaft of the Bunsen Coal Co., Danville, Ill., is divided into three compart-

ments and lined with concrete. One of the compartments, about 5 × 11 ft., is reserved for a manway and fitted with a steel stairway. The shaft is vertical, rectangular, 11 × 25 ft. inside lining, and about 210 ft. deep. The stairway, as shown in Fig. 139, is of zigzag pattern and, together with the landings, occupies a space longitudinally in the shaft for a width of 5 ft. 6½ in. Each flight rises on a 39° 25′ angle from the horizontal, and is 8 ft. long. The separate flights, including the landings, are supported on 8-in. channels weighing 11¼ lb. per foot, placed crosswise in the shaft, which in turn are supported at one end on the 10-in. I-beam dividers, while the other end is secured in the concrete end wall and has a bearing of 6 in. The channels are placed 2 ft. 5½ in. from the side walls, and are spaced vertically 5 ft. center to center, alternating for each flight. The stair stringers are ¼ in. thick by 7 in. deep. The treads are ¼ × 11 in. by 2 ft. 6 in., with checkered surfaces, and are supported on 1½ × 1½ × ¼-in. angles, riveted to the stringers. The rise between treads is 7½ in. The landing plates are ¼ in. by 2 ft. 9 in. by 5 ft., and are also checkered on the surface. Three lugs of 3½ × 5 × ⅜-in. angle iron are riveted to the landing plate and fastened to the concrete wall by means of ⅝ × 5-in. expansion bolts well drawn up. The hand railings are made up of two lines of 1½-in. pipe; the uprights are bolted to the stair stringers and connected to each line of railing. The total weight per vertical foot of stairway and landings, not including the channel supports, is 100 lb. The stairway is easy for the men to walk on and is of economical design.

V

DRIFTING

General Methods—Drilling Rounds—Support of Workings

GENERAL METHODS

Driving the Sheep Creek Tunnel.—The Sheep Creek tunnel was driven by the Alaska-Gastineau Mining Co., near Juneau, Alaska, to provide ore transportation facilities from mine to mill. Of the total tunnel length, 474 ft. was through slide rock and gravel near the portal, 4009 ft. in greenstone, 4224 ft. in slate and 1085 ft. in metagabbro; the last three rocks alternated, the stretches varying from a few feet up to several thousand. In general, the direction of the tunnel followed the strike of the formation, making it difficult to break the rock, especially the hard silicified slate, where the effect of the cleavage planes was marked. The holes broke short and a great many were required. Greenstone also developed cleavage planes in places, with a similar result. When the tunnel was in homogeneous blocky greenstone, the rate of progress jumped to about a foot per hour. In general, shallow rounds giving an average advance of 4 ft. were found most economical and rapid.

The tunnel was 10 ft. wide and 8 ft. high, with a small ditch along one side. Timbering was necessary only in the loose ground near the portal, and at this point the section was kept 8 × 10 ft. inside timbers. While the total length of tunnel is 9792.2 ft., from the solid rock face, where the main tunnel started, to its intersection with the crosscut from the Perseverance shaft the distance is 8800.5 ft. The tunnel was driven for single track its entire length, with the intention of widening out for sidings later. The average up-grade from the entrance was 0.65 per cent.

The force consisted of 70 men. Working on day shift only, there was a general foreman, a time-keeper, a tool sharpener, a tool-sharpener helper, a blacksmith, a blacksmith helper, a carpenter, an electrician, a powderman, and an outside man. Two compressor men worked 12-hr. shifts each. Divided into three shifts, there were 3 shift bosses, 12 upper-bar machinemen, 12 lower-bar machinemen, 18 muckers, 3 carmen, 3 locomotive engineers, 3 locomotive brakemen and 4 pipe- and trackmen. The arrangement of shifts was unusual. The cycle was completed in 18 instead of 24 hr.; during this time each of the three crews into which the force was divided worked 6 hr. and rested 12, so that

in each 24 hr. there was 8 hr. of work for each man. The incoming shift would relieve the outgoing shift at the working face and there was no intermission in the work for the purpose of eating a meal; this eliminated the delay of the meal time and the slackening of work following heavy eating.

After spitting the round, the men walked back in the tunnel about 1000 ft. The holes were counted and immediately after the last lifter went off the men started toward the face again. The fans being in full operation continuously, no smoke was encountered until within about 60 ft. of the face. One man then carried to the breast a water hose, previously connected to the water line, and sprayed the muck. Meanwhile the manifold was connected to the air line and the air hose attached thereto. The top bar was next brought up and jacked into place; two machines were placed on the bar, the air hose connected as soon as possible and drilling started. At the beginning of tunneling operations it required about 30 min. from the time the last lifter was fired until the top-bar machines were running. By the end of the undertaking this took frequently only 10 min. and seldom more than 15. With the top machines in operation, the four lower-bar machinemen started mucking out to make room for the lower bar. They threw back to the slick sheet, from which the muckers loaded into the cars. It was usually possible to set the lower bar and finish drilling the lower holes by the time the top machines were finished, so that all of the machines were torn down together.

While the machines were being placed in position, mucking began. The empty car was brought ahead and the track cleaned up to the slick sheet. Of the six muckers, four were shoveling continually and two resting. The average output of the force was 10 to 12 tons per hour. On a slick sheet beside the track, within 50 ft. of the face at all times, an empty car was kept. Whenever a car was loaded, the carman took it down the track and brought back the empty. There was thus only a fraction of a minute lost in changing cars. A few hundred feet back from the face another slick sheet was maintained and when the locomotive brought in a train of empties, they were thrown off the track on this sheet. The loaded cars were run back to this point, made up into a train and hauled out. From this latter sheet, the empty cars were run one at a time up to the sheet near the face. The empty cars weighed about 1100 lb. and were handled by a man using a crowbar. The sheets were kept about the same height as the top of the rail so that the car required to be lifted only the height of the flange. No switches were used anywhere. At no time did the removal of the muck limit the speed of drifting or interfere with the drilling cycle. The steel plates used for the slick sheet sidings were $\frac{1}{4} \times 48 \times 140$ in. The siding some distance from the face was 48 to 60 ft. long; directly opposite it were $\frac{1}{4} \times 22 \times 140$-in. sheets,

resting on planks down the center of the track, and also flush with the rail heads.

After practically all the muck was cleaned out, the shovelers laid ties to grade as the tunnel advanced, thus facilitating the subsequent laying of the rails. A 25-ft. false track was advanced over the ends of the rails as necessary, so that the cars were always next to the muck pile. When the advance permitted the insertion of a set of rails, the foreman and muckers did the work in 15 to 20 min., without disturbing the machinemen. On the completion of the round, shovelers, machinemen and everybody helped to tear down the machines and bars. Then $\frac{1}{4} \times 36 \times 140$-in. slick sheets were spread out for 30 ft. back from the face and covered with a little muck. These sheets were handled with grappling hooks through holes in each corner. When the machines were removed, an air hose was attached to a blow pipe and the holes cleaned. The shift boss, machinemen and foreman then did the loading. Spitting the fuses completed the cycle. There is no relation whatever between the shift and the drilling cycle, the machinemen relieving each other without stopping drilling.

Beginning with the time the last hole was heard to explode, the average time schedule for the last five months' operations would be about as follows:

Operations	Time consumed, minutes
Returning to face	4
Setting top bar	6
Mounting and starting up two machines	3
Shoveling back for bottom bar	30
Drilling 20 to 25 holes	210
Tearing down machines	5
Blowing out holes	4
Loading holes	6
Cutting and spitting fuses	2
Interval to report of first hole	4
Interval from report of first hole to report of last hole	3
Total	4 hr. 37 min.

In drilling the round the standard center cut, generally with six holes, was used; these cut holes varied from $7\frac{1}{2}$ to 8 ft. The side holes and lifters, which averaged $5\frac{1}{2}$ ft., were put in to suit the ground. Variations in the nature of the information necessitated a good deal of variation in the layout of the holes. A good many relievers and "kickers" were frequently necessary. The usual round required 21 to 23 holes, although at times 29 were necessary. For a 22-hole round the total footage was about 140; the average rate of drilling dry holes was 6 ft. per hour, of drilling wet holes, 7 to 10 ft. per hour.; the average time for a complete round was $3\frac{1}{2}$ to 5 hr.; the average advance per round was about 4 ft.

The muck from the face was hand trammed at first; subsequently a storage-battery locomotive was used, transporting material, men and waste rock. This was a 4-ton Jeffrey machine, equipped with 63 Edison, A-8, nickel-steel cells. The Matheson side-dumping roller-bearing cars used had a capacity of 30 cu. ft. Drill steel and miscellaneous material were handled principally on small flat cars. The 30-ft. capacity cars were the largest that could be used on the slick-sheet sidings. They were loaded heaping full, since, when loaded, they did not require to be derailed. The track gage was 24 in., the widest for which the side-dump cars could be built and allow the loaded car on the track to pass the empty on the siding. At the beginning of operations, the locomotive hauled out 12 cars in a train, but as the length of the tunnel increased, it became necessary eventually to handle trains of 30 cars. The material was used for fills on the surface railroad between the tunnel portal and the mill site, the filling being dumped from temporary trestles constructed for the purpose. One motorman and one brakeman handled and dumped the trains.

An exhaust ventilating system was used. The first fan was placed just outside the tunnel, exhausting 3000 cu. ft. of free air per minute; at 3000 ft. in a second fan was installed in series to handle the same amount of air; at intervals of about 2550 ft. additional fans were installed. A 15-in. ventilating pipe was made up of 18-gage galvanized iron in 25-ft. lengths. The lengths were provided with slip joints with lugs for wiring them together; all joints were wrapped with tarred canvas to insure their being air-tight. The ventilating pipe was carried on 4 × 6-in. vertical posts spaced 15 ft. It was kept close to the face and protected by a bulkhead of ties. A compressed-air line was carried along the bottom on 4 × 4-in. sills, which also supported the board walk; the end of this pipe was protected by the same bulkhead. The main line was always thus within a hose length of the face. The arrangement is shown in Fig. 140.

The rails were 50 lb., laid on 6 × 8-in. by 6-ft. ties. Six wires were carried the length of the tunnel; three of these were used for the single-phase, 110-volt lighting system, the purpose of the third wire being to give equal voltage at all points in the tunnel. The other three wires constituted the three-phase, 440-volt alternating circuit for the ventilating-fan motors.

Drilling was started November, 1912, but the work was not considered as completely organized until December, 1912. Between Dec. 1, 1912, and April 1, 1914, a period of 16 months, the tunnel was advanced 8707 ft. with a single heading, an average of 544.2 ft. per month. During the last six months, the average monthly advance was 596 ft. The greatest monthly advance was 661 ft. made in November, 1913. At various times advances of 24 ft. per day were made.

On account of the peculiar shift arrangement, the men were paid by

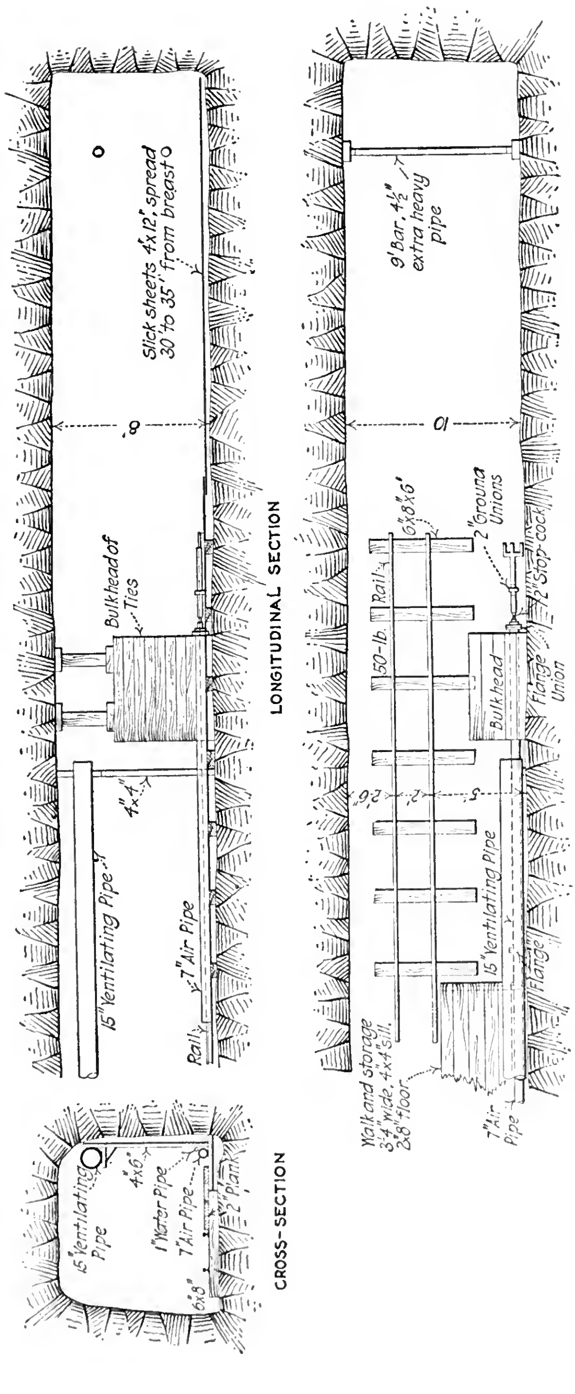


FIG. 140.—ARRANGEMENTS AT THE TUNNEL HEADING.

the hour, and in addition a bonus was distributed. The foreman distributed this according to his judgment. His idea was to give the greatest reward and the greatest incentive to the men chiefly responsible for the rate of progress. The result was thoroughly satisfactory.

The following cost data apply to the 8707 ft. driven between Dec. 1, 1912, and April 1, 1914, this representing the distance driven under the standard conditions described:

Wages	$14.72
Bonus	4.14
Explosives	4.47
Lighting	0.28
Tool replacement	1.35
Lumber and miscellaneous supplies	0.75
Store expense and transportation	0.38
Power and compressed air	2.53
Loss on boarding house	1.17
Depreciation on mining tools	1.29
Total	$31.08

DRILLING ROUNDS

Leyner Drilling Rounds (By Charles A. Hirschberg).—Reciprocating drills are restricted in their application to certain drilling rounds, while mounted hammer drills, such as the Leyner, are readily applicable to the drilling of all kinds of drift rounds. The piston or reciprocating machines are unsuitable for rounds of holes calling for drilling close to the top or side walls and at a slight pitch; such machines require ample head and wall room for operation; consequently they are slow in operation and hard to handle.

Fig. 141, at 1, shows a round of holes, to which the term "Leyner cut" has been applied, used in a drift or tunnel where the rock is extremely hard. It has been used with variations in the mines of Arizona, Colorado, and Michigan. It involves a pyramid or center cut including a great many upper or dry holes. The advantages of the Leyner cut are: (1) The holes are drilled with as few changes of machines and set-ups as possible; (2) a pyramid-shaped wedge of rock is first pulled from the center, after which the rest of the round breaks readily. While there is no hard-and-fast method of putting in such a round of holes, the principle is the same in all cases and involves the many upper and dry holes shown in the illustration.

Referring to the illustration, *C* designates the position of the crank of the drill in each case. *A* is a crossbar in the first position. From the top of the bar the four back holes, 9, 10, 11 and 12, are drilled. The machine is then "dumped" or tipped forward until the crank can just

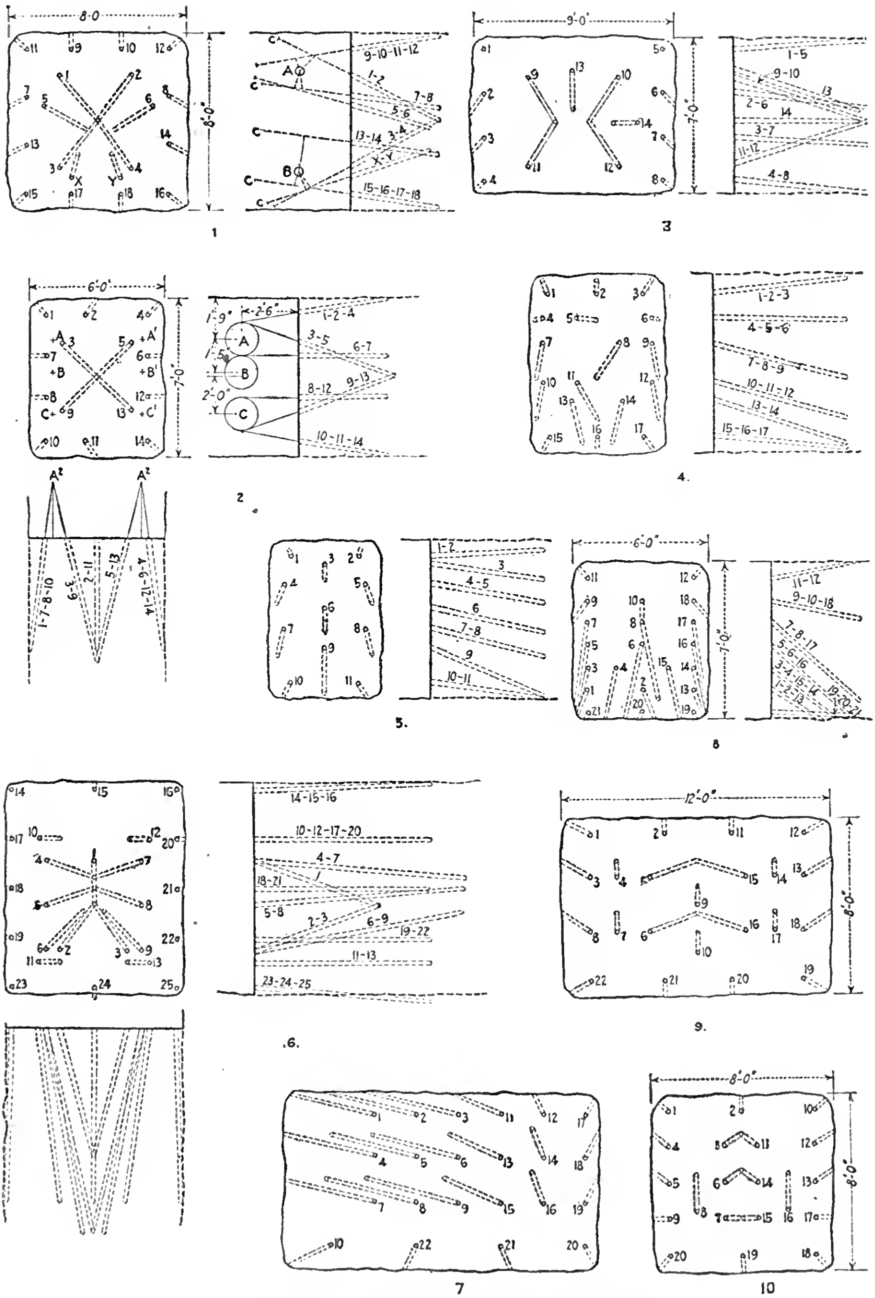


FIG. 141.—VARIOUS LEYNER DRIFT ROUNDS.

turn and clear the back or top of the drift, is moved out a little on the bar and the top center-cut holes 1 and 2 are drilled. If the bar is set up correctly in a drift of the size shown, the machine can be dumped enough to reach the center of the drift heading with the bottom of the hole. The machine is then turned under the bar and the side holes 7 and 8 and the cut holes 5 and 6 are drilled. The crossbar is next dropped to position *B*, the machine is set up on top and the side holes 13 and 14 drilled. Finally, the machines are turned under the bar, tipped up in front so that the crank just clears the bottom of the drift and holes 3 and 4 are drilled so as about to meet 1 and 2 in the center of the heading. The four lifters, 15, 16, 17 and 18, are drilled last, except that, if there is time enough, relievers something like *X* and *Y* may be put in to make sure. When sufficiently strong explosive is used. however. the round will break without these last.

The full round shown in the illustration is designed for hard rock, but a modified round of this kind could be used almost anywhere. In softer and better breaking ground, cut holes 5 and 6, relievers *X* and *Y*, one lifter and one back hole can be left out, but the four cut-holes 1, 2, 3 and 4 are nearly always used and are pitched up or down, and in, to meet about at the center. There are two reasons why this round is practically impossible with piston drills. First, they cannot drill it fast enough, particularly on account of the dry holes; and second, the size of the piston drill is too great to permit operating it in the positions necessary to give the holes the proper pitch and angle.

At 2 is shown another pyramid cut as used in some of the mines of Mexico for driving small drifts in hard rock. This round is drilled from an arm mounted on a column, *A* and *C*, A^1 and C^1 representing the vertical positions of the arm, and A^2 the horizontal position of the arm and drill on both sides of the column; the column is placed midway between the walls of the drift. Holes 4 and 5 are first drilled from the top of the arm, on the right-hand set-up. The drill is then swung under the arm and hole 6 put in. Next the arm is swung to the left-hand side of the column and hole 7 drilled. The machine is turned to the top of the arm and holes 1, 2 and 3 are drilled. The arm and drill are then dropped to *C* and hole 8 drilled; the machine is swung under and 9, 10 and 11 put in. Holes 13 and 14 are drilled by swinging the arm and drill to the right of the column with the machine underneath. The machine is turned on top of the arm and hole 12 drilled, which completes the round. For extremely hard ground extra holes may be drilled with the arm and machine at *B*, but in all moderately hard rock this has not been found necessary.

A round of holes employed in the mines of South Africa, in a drift 9 ft. wide by 7 ft. high, is shown at 3. It usually comprises 12 holes. Hole

13 is sometimes drilled when the rock is not breaking properly, while both 13 and 14 are used when extremely hard rock is encountered. The distance between holes 1, 2, 3 and 4 in the vertical line is approximately 2 ft., likewise the distance between holes 5, 6, 7 and 8. Holes 9, 10, 11 and 12, or, in other words, the cut holes, are put in approximately 4 ft. apart at the face of the rock, but holes 9 and 10 slant downward and inward and meet holes 11 and 12, which slant upward and inward. The distance between the junction of holes 9 and 11, and 10 and 12, at the bottom, is approximately 18 in. Hole 13, when used, is put in 1½ ft. below the top of the drift and slanting downward until it comes to about a central point 18 in. from the junction of holes 9 and 11 and from that of 10 and 12. Hole 14, when used, is put in at the face, about 2 ft. 6 in. from the center of the cut, and slants in as shown to a distance of about 18 in. from the junction of holes 10 and 12. Usually, however, holes 13 and 14 are not used. The round of 12 holes generally breaks between 5½ and 6 ft. of ground. The machine is mounted on a column and arm.

A round used in the Cripple Creek district of Colorado is shown at 4. It consists of 17 holes and is used only in drifts 8 ft. high by 6 ft. wide or larger, where the rock is an exceedingly hard phonolite. It will be noted that with the exception of the back holes 1, 2 and 3, all the holes point downward. This round will break between 5 and 6 ft. of ground. The same round is shown at 5, modified for a smaller drift, one 7 ft. high and 5 ft. wide. It consists of but 11 holes and is used for drilling in brecciated formation and in vein matter. The ordinary double-screw column with one set-up is used for both rounds. Of course, the arm is shifted from side to side and lowered as occasion requires, the holes being drilled from both above and below the arm. These rounds are varied slightly with the nature of the ground; fewer holes are sometimes drilled, but never more.

A round used several years ago in driving the Lucania tunnel at Idaho Springs, Colo., is illustrated at 6. This tunnel is 9 ft. 6 in. high by 8 ft. wide, and the advance averaged between 7 ft. 6 in. and 8 ft. per round. The set-up involved the use of two columns, one carrying two arms and the other one arm, making a total of three machines. Short cut-holes 1, 2 and 3 were drilled 6 ft. deep; long cut-holes 4, 5, 6, 7, 8 and 9, 9 ft. 6 in. deep; relievers 10, 11, 12 and 13, 8 ft. deep; back holes 14, 15 and 16, 8 ft. deep; side holes 17, 18, 19, 20, 21 and 22, 8 ft. deep; lifters 23, 24 and 25, 8 ft. deep. Holes 6, 2, 3, 9, 11, 13, 23, 24 and 25 were drilled by the bottom machine; the rest were drilled by the two top machines. The round was shot in the order numbered, the two cuts being loaded and fired first. The remainder of the round was then loaded and fired.

A round employed in the Michigan copper country for what is known as a drift stope is exhibited at 7, the width of the working varying according

to the width of the lode; the object is to take out all of the rock between the foot and hanging walls, the holes being pointed in some cases toward the foot and in others toward the hanging. The round as illustrated would do for an 8 × 14-ft. drift. The set-up consists of a double-screw column with arm.

At 8 is shown the method in use at the Quincy mine for small drifts; it works satisfactorily in this particular case owing to the fact that the driving is done entirely through trap, there being no copper to contend with. This round is not good, however, in ground that is heavily charged with copper, since great difficulty would be encountered in getting the cuttings out of the holes. In such cases the direction of the holes should

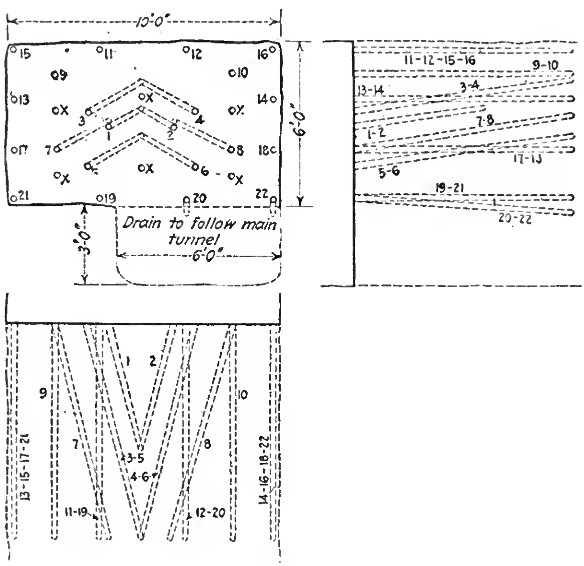


FIG. 142.—SUCCESSFUL ROOSEVELT-TUNNEL ROUND.

be reversed so as to point upward. The round is shown for a 6 × 7-ft. drift and is drilled from column and arm. The holes are drilled to the following depths: 1, 2 and 3, 2 ft. 6 in. deep; 3, 4, 14 and 15, 3 ft. 6 in. deep; 19, 20 and 21, 4 ft. deep; 5, 6 and 16, 4 ft. 6 in. deep; 9, 10, 11, 12 and 18, 5 ft. 8 in. deep; 7, 8 and 17, 6 ft. deep.

At 9 is illustrated a round of holes for an 8 × 12-ft. drift as used in the Dober mine near Iron River. The ground was a gray slate. The holes were drilled from a column and arm and required two set-ups, owing to the wideness of the drift. All holes were drilled to a depth of 5 ft. with the exception of those numbered 5, 6, 15 and 16, which were drilled 5 ft. 6 in. deep.

A round employed in medium-hard iron ore at the Cary mine, Hurley,

Wis., is shown at 10. The size of the drift is 8 × 8 ft. All holes with the exception of 18, 19 and 20 look up a little above the horizontal. Holes 1, 2, 4, 5, 9, 10, 12, 13 and 17 are drilled 5 ft. deep; 18, 19 and 20, 5 ft. 6 in. deep; 3, 6, 7, 11, 14 and 15, 5 ft. 8 in. deep; 8 and 16, 6 ft. 6 in. deep. This cut breaks well and lengthens the drift 4½ ft. with each round. The round is drilled with column and arm set-up.

After the trial of several systems of placing the drill holes for the Roosevelt tunnel, that shown in Fig. 142 finally proved to be best adapted to the tough nature of the jointless rock. Water-Leyner drills were employed. In attacking the ordinary rock, all holes were drilled 8 ft. except the cuts and relief cuts, Nos. 1 to 8, inclusive, which were drilled to a 10-ft. depth. In tougher ground, these depths were each cut down 2

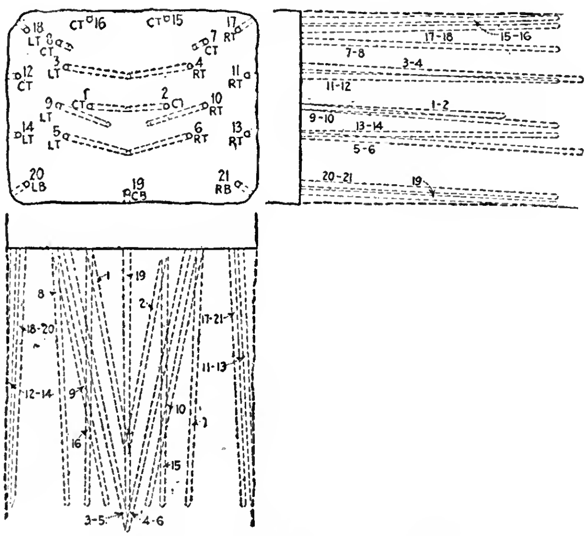


FIG. 143.—LARAMIE-POUDRE ROUND.

ft., and in addition to the 22 holes used on the ordinary rock and numbered in the illustration, the six extra holes X were put in. At first, even with the use of from 300 to 350 lb. of 60 per cent. dynamite, great difficulty was experienced in properly blasting the eight cut-holes, sometimes several loadings being necessary to blow out the cut. Finally, however, after putting in the two extra cuts shown, even the toughest ground yielded. The system of placing the holes was evolved with a view not only to blasting the rock to the best advantage, but also to allow the greatest economy of time in drilling. These ends proved to be best effected by mounting the two Leyner drills on a single, horizontal crossbar, instead of on the more usual two independent vertical columns. In this way even the maximum number of 28 holes required but two set-ups

of the bar. The grade line was carried about 18 in. below the top of the bore and about 8 to 12 in. below this was placed the bar. From this, the center and corner back-holes were drilled, and then by revolving the drill around and beneath the supporting bar, all of the remaining holes except center lifters and bottom corners were put in. From the second position of the bar, usually about 18 to 24 in. above the floor, the last four holes were put in. In tough ground an extra center-cut hole X was also put in from this set-up.

The Laramie-Poudre tunnel at the beginning of the year 1911 held the best two American tunnel-driving records: 609 ft. in January, 1911, and 653 ft. in March. Fig. 143 shows the layout of the holes in regular work. The holes were drilled and shot in the succession numbered in the cut, requiring two set-ups of the tunnel bar, no column being used. The upper set-up was drilled on top of the muck pile, and in the meantime the muck was cleared away, when the bar was lowered and the lifters put in. Holes were started 2⅛ in. in diameter and bottomed at 1⅜ in. Two water Leyner machines were used, drilling 10-ft. and 12-ft. holes. In case of extremely hard rock a third machine was mounted, each machine drilling holes as follows: Those lettered LT were drilled by the left-hand machine on the top set-up, those marked CT by the center machine on the top set-up, and those marked RT by the right-hand machine on the top set-up. The bar was then lowered and each machine put in a lifter, lettered LB, CB and RB. The blasting charge for a round generally consisted of about 100 sticks of 100 per cent. gelatin, 150 sticks of 60 per cent. and 250 sticks of 50 per cent.

Drift Round in Flat Sediments.—In Fig. 144 is shown the side-cut drift round used in southeastern Missouri for drifts in the flat-dipping dolomite that forms the country rock. The drifts are about 6½ ft. high and 8 ft. wide. Other types of rounds have been tried. In one of the mines where Leyner drills are used, the Leyner center, pyramidal cut has been tried, but it did not prove so satisfactory as the side cut. The Western cut, in which the holes are drawn from the bottom with lifters to take up the bottom bench lift under the cut, has also been tried, but the side-cut has held its own.

Only at the mines of the Desloge Consolidated is there used a different cut. There, when drifting in ore, a center V-shaped cut is used in the drift, and a drift 11 ft. wide and 6½ ft. high is driven. This, it is stated, can be advanced as fast as the smaller drift. Two one-man machines drill 18 holes to the round making a total drilling of about 100 ft. A round will square up about 5 ft., while the average speed is 90 to 100 ft. per 25-day month. About 50 lb. of dynamite breaks the 25 to 30 tons of rock coming from the round. The only extra expense is in handling the rock and when the work is in ore this is broken much more

cheaply owing to the much smaller amount of dynamite used per ton broken.

The reason the side-cut proves so satisfactory in this sedimentary formation is that the holes, owing to the flat dip of the limestone, are always kept in the same bed. Consequently, they are easier to drill and break better from being in rock of the same character for their entire depth. Where drilled downward there is frequently a tendency for the holes to "bull-ring," breaking out in the weaker beds and leaving the more resisting beds standing. This causes the holes that follow to break poorly, and the entire round is often held up.

Two one-man piston drills are usually used in a drift, one mounted

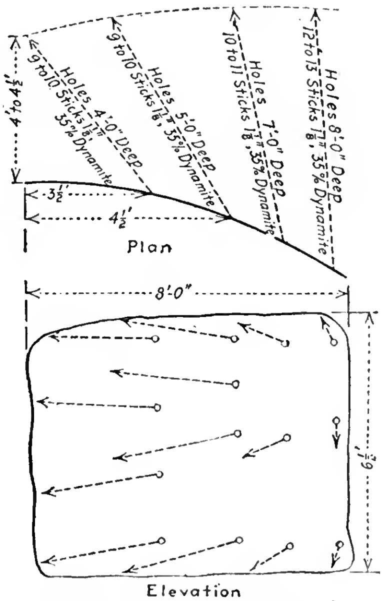


FIG. 144.—SOUTHEASTERN MISSOURI SIDE CUT FOR DRIFTS.

on each side of a vertical column. With two drills a round can be drilled and blasted in one shift. In drifting the drilling is done only on day shift, and the machine men always have a clean face to start with. A one-man piston machine will drill about 40 ft. of hole in average ground, taking out the time required in setting up and loading and blasting. A Leyner drill has often been known to put in a drift round in a shift with one man running it, in the mines where that type of drill is used.

With this side cut, the round is alternated from one side to the other. A round will square up from 4 to 4½ ft. per shift, and the average speed of drifting in the district is about 90 ft. per month of 26 days. Generally from 11 to 14 holes will break a round as shown in the accompanying dia-

gram. About 20 lb. of dynamite per foot is used in such a drift and from 12 to 18 tons broken by a round. The cost is about $10 to $11 per foot, of which about 25 per cent. is for labor, this cost covering the direct charges such as breaking, shoveling, tramming, drill upkeep. air and explosives.

Rapid Drifting by St. Joseph Lead Co.—The St. Joseph Lead Co. connected its No. 1 and No. 7 mines at Bonne Terre, Mo., with a drift 7 ft. high by 12 ft. wide in hard unstratified limestone. A record of 455 ft. was made in 25 working days of three shifts each, or 75 eight-hour shifts. The force consisted of three machine men, three chuck tenders, two muckers and one mule driver for each shift. Three Sullivan 2¾-in. piston drills were used; the air was supplied under 90-lb. pressure. The

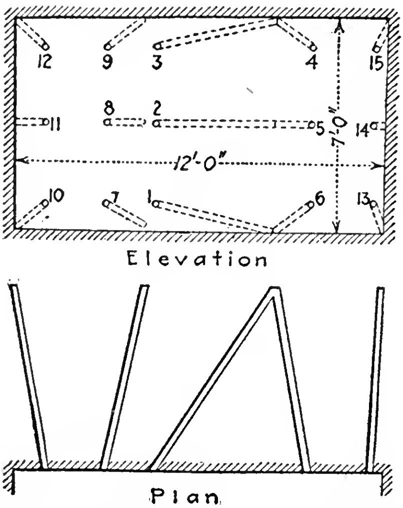


FIG. 145.—FIFTEEN-HOLE ROUND FOR WIDE DRIFT.

drilling was done from a crossbar to which were attached three arms similar to those used on columns. This enabled the miners to work the machines one above the other when necessary, and if one side of the drift was harder than the other, one of the machines could be moved over to the hard side so as to drill in any position desired, without interfering with either of the others.

To pull a 6-ft. cut 15 holes were necessary, placed as shown in Fig. 145. The cut holes were drilled to meet at the points. A vertical row of three relief holes widened out the cut and the six side holes finished squaring the face. When the ground was harder than usual, eight, instead of six cut holes were used. Each of the cut holes, 1, 2, 3, 4, 5 and 6 was charged with six to eight 1½ × 8-in. cartridges of Forcite 60 per cent. gelatin dynamite. Each of the remaining holes received three, four or five

cartridges of the same explosive, according to the character of the rock encountered, making 80 to 90 cartridges, 62 to 72 lb., for the entire round. This gives an average of 11.2 lb. of explosive per foot, or 1.44 lb. per ton of rock.

The blasting was all done by electricity. Victor electric fuses were

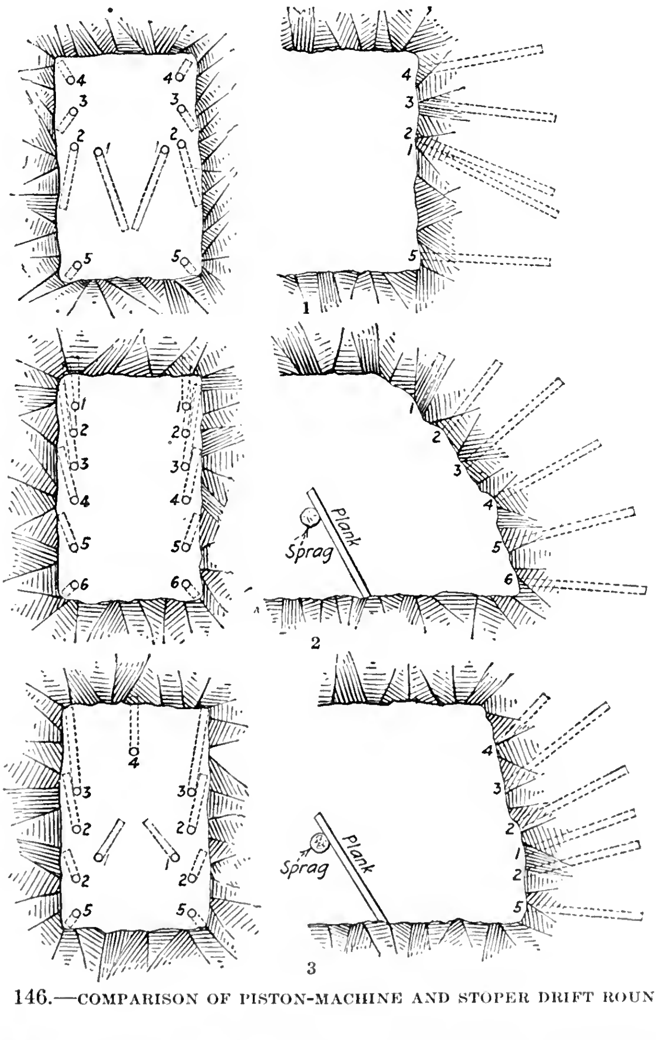


FIG. 146.—COMPARISON OF PISTON-MACHINE AND STOPER DRIFT ROUNDS.

used. The No. 5 were discarded for No. 8 on account of the better results in the work done by the explosive and in the character of the fumes after blasting. The blasting machines used were the Reliable No. 3 and No. 4. The lead wire was 500 ft. of Duplex No. 14 B. & S. gage. The blasting was done from small chambers cut in the side of the drift. The drilling,

including setting up and tearing down, took six hours. The blasting occupied about one and one-half hours.

Drifting with a Stoper (By G. E. Wolcott).—The stoper has, through the requirements of the leaser and small operator, been adapted to drifting. In the case of the leaser, low equipment expense is imperative. It is this fact rather than any advantage of the machine itself that is responsible for its use for drifting.

In the Cripple Creek district the drifts are usually from $3\frac{1}{2} \times 6\frac{1}{2}$ ft. to 5×8 ft., and the arrangement of holes for a piston machine in average ground is generally as shown in Fig. 146. When the stoping machine is used for drifting, holes are placed differently. Since flat- and down-holes are drilled with difficulty by this machine, as many holes as possible are inclined upward, and as a rule only the two lifters are drilled looking down. In 2 is shown the general arrangement of holes. A set-up is made by using a plank leaning against a sprag as shown. The spud of the machine is placed against the bottom of the plank for drilling all holes except the lifters; for the latter it is raised a foot or more, as desired. The face of the drift is not ordinarily kept vertical, because the holes are more readily drilled with an inclined breast as shown. The holes do not, however, have a good show to break to start the round, and in order to keep the back to the required height it is necessary to drill the holes deeper than they will break. A better arrangement, and one requiring fewer holes, is shown in 3, in which the cut holes are bottomed as near the center of the breast as possible, where they have the best chance to break. With the stoper it is impossible to obtain so good a cut as with a piston machine, and in hard-breaking rock this must be offset by an increased number of holes.

A drawback to the use of the stoper for drifting is the fact that the muck must be thrown back farther from the breast to make room for the machine. The ordinary stoper requires not less than 6 ft. between the spud and the breast. If the stoping bar is shortened, too many changes of steel are necessary.

Recording Mine Timbering (By John T. Fuller).—The method of posting mine timbering here described refers especially to "gangways" or drifts, but can easily be extended to embrace all timbering, such as that of shafts, etc. It is difficult to keep track of the timbering in a large mine. The timber in any particular tunnel is put in, as a rule, at different times as the work progresses; retimbering is perhaps of frequent occurrence and the age of the timbering or of any particular set of timber, the cause of failure and many other items of information that are of value to the mine manager soon become hopelessly confused and lost. The method of computing the cost of timbering at many mines is simply to charge against this item the timber sent underground and the labor involved in preparing and setting the same. Where timbering and retimbering are

done by contract and payment made once a month or once a fortnight it becomes absolutely imperative in fairness both to the company and to the contractor to adopt some system of recording or posting the timber sets. Unless some good system is in effect both the contractor and the head timberman, or other company men in authority, are likely to become confused and uncertain as to exactly what timbers have been set during the period in question. Especially is this true in the case of retimbering, which frequently means simply resetting timbers at irregular intervals.

In one case it was attempted to keep track of the timbering by having a special set of tracings made of each level in the mine. When the engineers posted the development work at the end of each period they also posted the timbering with the aid of the head timberman. Each set was then plotted on the tracings in its proper place. So far as keeping track

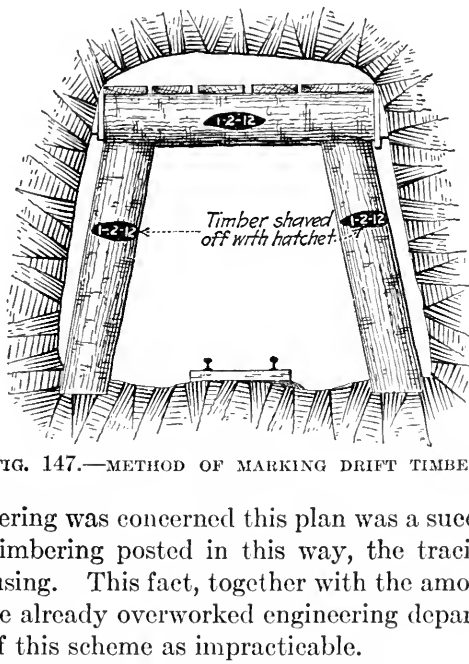


FIG. 147.—METHOD OF MARKING DRIFT TIMBERS

of the new timbering was concerned this plan was a success, but in trying to keep the retimbering posted in this way, the tracings soon became hopelessly confusing. This fact, together with the amount of extra work thrown upon the already overworked engineering department, led to the abandonment of this scheme as impracticable.

The problem was finally solved as follows: A set of 11 steel figures, including a spacer, was ordered. Every stick of timber in the mine after being set in position was blazed with an ordinary carpenter's hatchet and the date of setting punched thereon with the steel figures, and then painted over with one coat of a wood preservative paint as indicated in the accompanying drawing. In the drawing, Fig. 147, the blaze mark and the date are exaggerated for the sake of clearness.

The head timberman on his final round just before the end of the shift, took with him, in a canvas bag slung over his shoulder, the steel figures

RE TIMBER SHEET .... Sheet No. 1 ....

Level .... Date .... Shift ....

Location	Caps		Posts		Sills		Cost of Timber	Labor Cost Erection			Total Cost	Date Old Timber Put In	Life	Cause of Failure
	No.	Dimension	No.	Dimension	No.	Dimension		No.	Rate	Cost				
					Lagging									
						Wedges								

Head Timberman .... Chief Time Keeper .... Store Keeper ....

FIG. 148.—FORM FOR RECORDING RETIMBERING IN THE MINE.

required to mark the date of that day, a small hatchet, about two pints of paint in a specially stoppered can, a brush, pads of the "Retimber Sheet" form, shown in Fig. 148, and of another form called the "Timber Sheet" in which the columns marked "Date of old timber put in" and "Life" are omitted and the heading of the last column "Cause of Failure" is changed to "Remarks." He blazed and marked each timber as shown and entered the required data on the proper form, leaving black columns which are headed "Cost of Timber," "Labor Cost Erection," "Total Cost" and "Life." Before going to the next level he would leave the filled-out forms at the tool-house, where at the end of each shift the timekeeper found and collected them. The timekeeper filled out the column marked "Labor Cost Erection" and checked the labor items. On reaching the surface the timekeeper turned the forms over to the storekeeper, who filled out the "Cost of Timber" column and returned the forms to the timekeeping department where the remaining columns were filled in, the whole sheet checked over and delivered to the mine manager before the miners and other workmen in the mine coming off shift had changed their clothes and left the mine. It was thus possible for the manager to check up any questionable report with the head timberman and the men who had performed the work before any appreciable time had elapsed after its completion. The sheets were kept on file by levels. The various items as shown by the sheets were posted and totaled daily by the bookkeeper on the large mine-cost sheets, so that the end of a period entailed no extra work to find the amount due the contractors or the cost of the mine timbering.

The date mark placed on the timbers in the way described will remain clear and legible as long as the timber lasts. Old timbers reset were marked a second time, leaving the original marks intact so that when a set of timber was finally removed the individual pieces forming the set practically told their own history and age. Where the timbering is done by contract the columns headed "Labor Cost of Erection" are designated as "Cost of Contract" with subheads; "Sets," "Feet," "Rate" and "Cost."

Mesabi Underground Turn Timbering.—In underground mining on the Mesabi there are nearly as many different styles of timbering for drift and crosscut turnouts as there are separate operating companies.

On sublevels, which are nearly always temporary, turnouts and curves are put in by the foreman without help from the engineering department. A simple turnout commonly used in such cases is shown at 1, Fig. 149. Here three drift-set posts are taken out and replaced by a set called the open set. The only drawback in this method is the loss of headroom.

For main levels, the chief considerations are permanency and ease in tramming. A square turn, such as is used on the subs, requires so long a

cap to enable motors or mules to pass that it becomes weak and frequently causes a great deal of trouble as the ground takes weight. It has, therefore, become generally customary to put in a carefully designed curve of 25- to 40-ft. radius. The method of timbering and laying out the curve then depends on the captain or the operating company.

At 2, 3 and 4 are shown standard main-level curves used by one of the large operating companies. The lines *AB* are established by the engineer by putting two nails in the caps. Then the foreman by use of the

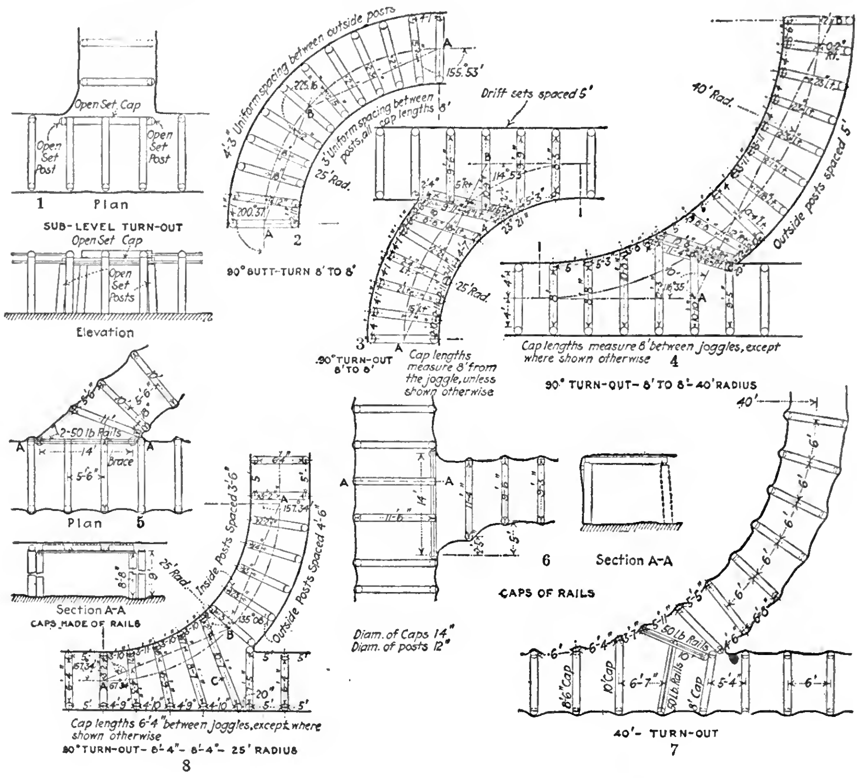


FIG. 149.—VARIOUS METHODS OF TIMBERING TRACK TURNS ON MESABI HAULAGE LEVELS AND SUBLEVELS.

blueprint is able to place accurately each set. At 5, 6 and 7 are shown curves that make use of 50-lb. rails instead of long timber caps. These curves have the advantages of requiring less supervision and engineering, of being stronger, and of not requiring so high an excavation to secure equal headroom. At 8 is shown a curve frequently found in the mines of one of the larger companies. Although a good-looking curve for firm ground, it has the disadvantage of the long cap *C*, which is liable to break before the drift is ready to be abandoned.

Building Drywalls, Sudbury District (By Albert E. Hall).—In the Sudbury nickel district, although a new method is to replace the drywalls, it is still customary at present to drive a small stope and follow up with two drywalls over which lagging is placed to form a drift. The machines are then started breaking ground over the walls. Care is always taken to load the walls evenly, so that there will be no tendency toward movement as the result of unequal loading. Chutes are built in every 30 ft., and the even loading is especially important at these points, as slight differences in weight will cause the chutes to move. The stopes are carried up on the shrinkage system. It is important that the walls be well built; any defects may cause serious trouble, expense and delay.

The walls are built so as to give between track and lagging a clearance convenient for walking. They are 6 to 7 ft. high and 4½ ft. thick. A

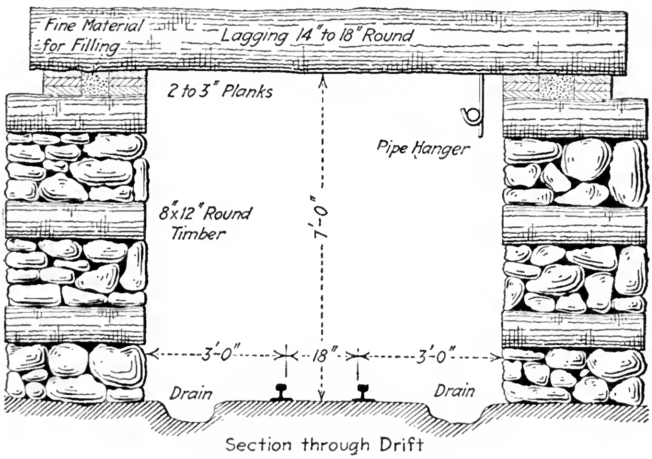


FIG. 150.—DRYWALL AND TIMBER FOR DRIFT.

course of timber is placed every 2 ft., and the timbers in each course are about 2 ft. apart. The timbers extend through the wall and serve as a bond to hold the wall together. As a further bond, numerous headers are put in. The walls are built on contract at 12 cts. per square foot of wall face. The company has to furnish the contractors with stone and timber, two trammers usually being assigned to the masons in the event that no stone is convenient. The walls on the old levels are pulled down so that the timber can be used over on the lower levels. On top of the walls are placed two rows of 2-in. planking spiked to the top course of timber in the wall, one behind the other, so as to give a large bearing surface for the lagging. Between the two rows of planks about 6 in. is left and is filled in with fine stuff. The method of construction is shown in Fig. 150. Care must be taken that only flat stones are put in the wall,

since any that are not flat cause either themselves or the adjoining stones to be squeezed out when the wall is loaded. Strictly speaking, drywalls are not advantageous where sufficient rock is not obtainable to do the building. This condition exists in the Sudbury district, *i.e.*, not enough rock is broken with the ore or in development drifting. For this reason some ore must be used, and since the ore has not sufficient crushing strength, some of the walls have failed where a piece of ore in the face was crushed.

At points opposite chutes a high wall is built, consisting of a square timbered crib filled in with rock between the timbers. Where a drift branches, the point is also made of cribbing, either square or round, and is filled in with stone. A wall is built around behind the chutes and in some cases mills are built above them. These are made round. The timber in these is laid in courses 2 ft. apart, as in the walls, but the individual timbers are placed touching each other. Manways are placed at suitable points and are built 5 ft. square. The timber in these is placed in a manner similar to that used in building up a mill.

VI

STOPING

Methods—Timbering and Substitutes—Filling—Various Devices

METHODS

Stoping at the North Star Mine (By L. O. Kellogg).—At the North Star Mine in Grass Valley, California, the ore occurs as a series of shoots in a flat quartz vein. The average dip is about 23°, the width about 2 ft., but a greater thickness is broken to get working room. The main inclined raise, through which material is lowered to the vertical hoisting shaft, has loading stations at three levels, spaced something over 300 ft. on the incline. To these bins the ore is trammed from both sides of the raise by mule or by hand, according to the length of the haul and the tonnage handled. The ore from the block above is got to each level by means of gravity planes. These are locally and aptly called "go-devils." They are peculiar in the fact that the same cars that are used for tramming in the stopes are lowered in balance on the plane by means of a light, portable and easily controlled triple-sheave head-block. A typical stope in process of working by the go-devil method is diagrammatically illustrated in Figs. 151 and 152. These exhibit, by plans in the plane of the vein, four successive stages in the working of a stope, the earliest at the top of Fig. 151, the latest in Fig. 152.

The inclined raises which are put up at intervals for development may or may not be in ore. A stope may be started between two raises, extended to each, and may use a go-devil in each raise, it may be started on both sides of a raise and may use a single go-devil in the center, or it may be started in unexplored ground and an entirely new go-devil installed near the middle. In describing the development of a typical stope, we may consider one started on two sides of a raise, which is itself in ore. Such a raise will have employed a go-devil, and the chute at its bottom and the tracks will have been left in place.

The first operations consist in drilling holes along the side of the drift. Usually two stoping points are selected, one on each side of the raise, and each stope carried toward the raise in one direction and to the limits of the oreshoot in the other. The first ore is shot down on plats, usually covered with steel sheets, laid over the level tracks. The stope soon assumes the shape shown at the top of Fig. 151.

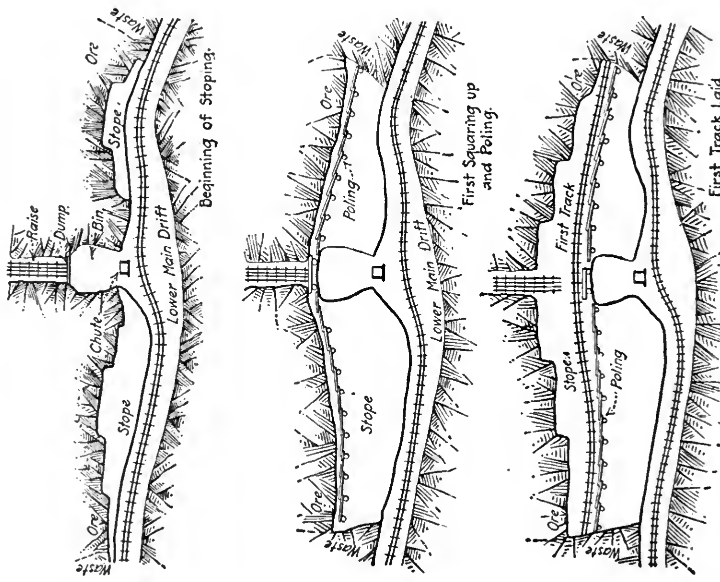


FIG. 151.—THREE EARLY STAGES IN THE DEVELOPMENT OF A TYPICAL STOPE.

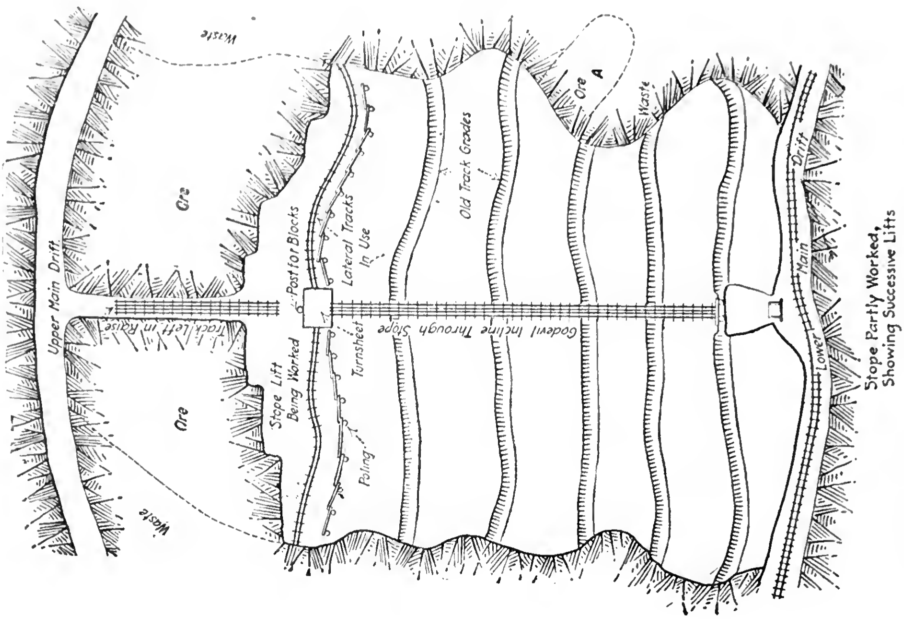


FIG. 152.—STOPE WELL ALONG IN DEVELOPMENT.

The foot-wall of the vein has usually been carried pretty well up on the side in drifting, so that after two or three cuts have been shot out, a shoveling plat can be erected, the height of a car or higher. Such a plat is illustrated at the top of Fig. 153, consisting in its simplest form of 2-in. planks, covered with steel, set at right angles to the track on a round timber supported by two stulls. In the same way, small chutes or lips can be put in to deliver directly to the cars without shoveling if the vein is steep enough.

In this manner the stope is carried up until its face is somewhat above the top of the raise chute. About 30 ft. on the dip may have been thus mined. It is squared up and the pillars next the chute are removed. Then a row of stulls is placed within about 5 ft. of the stope face and horizontal poles, with a minimum diameter of 6 in., are set against these so as to keep the broken rock on the intermediate level about to be started.

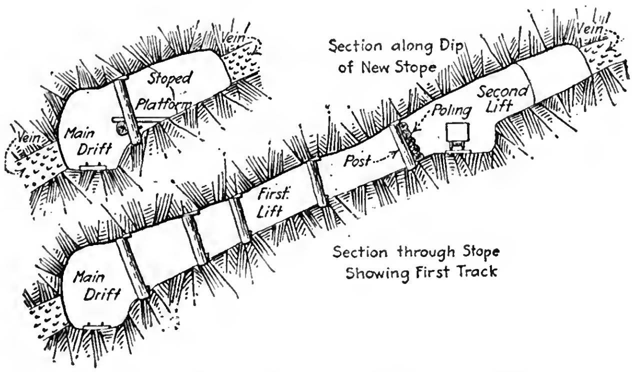


FIG. 153.—STOPE SECTIONS DURING EARLY OPERATIONS.

The face has previously been drilled up for its full length. Stopers are used for nearly all drilling in the stopes. When the stope is thus "poled," it appears as shown in the second stage of Fig. 151. The holes are blasted, beginning at the raise, and the ore is thrown toward the raise, whence it is shoveled into the chute. When this first cut has been carried so far that shoveling becomes slow, a Leyner or a Jackhamer is used to shoot up the foot-wall, making a level cut about 5 ft. wide. This foot-wall breaking is done with long "lifters" or with numerous "pops" about at right angles to the foot-wall; the "lifters" are also drilled with stoping drills if circumstances permit. On this flat grade, a track is laid both ways from the chute and the broken ore is handled with cars thenceforth. A cross-section of part of the stope, Fig. 153 below, shows the poling and track.

A number of cuts are started at the raise and carried back until the stoped portion along the raise measures from 25 to 30 ft. The stope may then look somewhat as in the third stage of Fig. 151. The ore is blasted

on steel sheets or on 2-in. planks, laid on the tracks until some distance is gained up the dip; then raised plats are put in as when stoping along the level.

The lift is then squared up by carrying back each cut to the limit of the ore; the face is drilled up, and poling again put in. At the same time the Leyner machine is used to break out a place for a turnsheet in the raise foot-wall at the level of the new poling and a turnsheet is placed there; a post to carry the go-devil blocks in the raise above the turnsheet is also set and the dumping device at the chute top. The first ore blasted is shoveled into cars fastened to the go-devil rope and then lowered and dumped. Stoping proceeds away from the raise as before and when the track is installed, the cars are handled with the go-devil. The tracks and ties from the lift below are used for the new lift.

This operation is repeated until the oreshoot in that block is worked out. Fig. 152 shows the appearance of a stope in which six lifts have been made. The economical width of each lift, measured up the dip, is calculable, being determined by comparing the cost of installing the new poling, track and turnsheet, including loss of time, with the cost of shoveling down from the face. The poling serves to keep the broken ore at or above the track level, and also acts as a partition between ore and waste, the coarse waste being thrown below, where it is available for building dry-wall supports to the roof when desired. It must not be supposed that the stulls of the poling are the only ones set. As a matter of fact, stulls are placed at frequent intervals throughout the stope as needed.

The advantages of the system as a method of handling material in a flat-dipping stope are numerous: (1) It reduces to a minimum the necessary shoveling. While it is still necessary, as each lift reaches its upward limit, to shovel the ore twice, this is nothing compared with systems where the muck may require shoveling over 100 ft. The shoveling is furthermore made easier by the use of plats where possible. (2) It eliminates much dead work. Except for the track grading and occasionally some main-level chutes, the work is all in the ore; no long crosscuts are run. (3) It has great capacity. Three cars in a stope will handle the product of four or five machines. Its capacity is especially noticeable as compared with systems involving shaking chutes, which were tried and discarded. (4) It consumes no power, as does a system involving hoisting from underhand stopes or from workings down the dip; nor is there any expense for a man on the hoist. (5) It is cheap in the final results and not so expensive in material as might be at first supposed, inasmuch as cars, ropes, blocks, turnsheets, trucks, ties and poles are used over again until worn out, and are often taken to other stopes. (6) It involves the minimum rehandling of material, except at the level, as compared with systems using a series of chutes down the stope with intermediate tramming. (7) It reduces the

number of main-haulage drifts, with a consequent reduction in development cost, first cost of bins and cost of tramming. (8) It has great flexibility. As can be seen, it can take care of a stope of any size or any shape. By sending out the lateral tracks at 30-ft. intervals, the limits of the oreshoot can be outlined exactly. If, in any case, the ore hooks down into the waste as shown at *A*, Fig. 152, it is only necessary to return to one of the lower levels, when the stope is worked out, and continue it as a small drift so as to tap the offshoot from below.

There are, however, certain disadvantages and limitations: (1) Sufficient headroom over the tracks is necessary to allow a go-devil car with a projecting rock to clear the roof. To obtain this means considerable expense in cutting out the foot-wall. The hanging is left intact as nearly as possible. (2) It will not operate below a 5° and with difficulty below a 10° dip, and conversely when 35° is reached, its operation becomes difficult and sometimes dangerous. (3) The slowing down of operations and decrease of output upon cleaning up, beginning a new lift, is a source of trouble and makes it difficult to keep up production if several stopes are "moving" at the same time. (4) There is some danger of delay from accidents; the cars on the incline may be derailed or run away. (5) A good hanging is a great advantage and almost a requisite.

In regard to costs: Where the stoping operations utilize the go-devil in an old raise, the cost of the go-devil construction is properly charged to the original raising. The installation of successive turnsheets, however, involves breaking and replacing the old inclined track to such an extent that little of it is actually used. The cost of laying go-devil tracks, as with most of the other of these costs, will vary exceedingly, according to the degree of uniformity of dip, accessibility of working, height of hanging wall, etc.; the labor cost varies from 60 to 80 cts. per linear foot. The cost of rail and timber for a new go-devil will average from 50 to 60 cts. The cost of the dump, bin, chute, etc., is similarly usually chargeable against the original raise and when new work must be done for stoping, conditions are so variable as to make figures valueless.

Poling is also subject to considerable variation, depending on the accessibility of the stope and on the width. The stulls are rounded on the foot-wall end and are set in hitches. The timbermen who do this work are paid at the rate of $3.25, the uniform shift being eight hours. Labor will vary from 10 cts. to 33⅓ cts. per linear foot. Material cost will vary according to the number of old poles employed. A rough average would be 25 cts. per foot along the stope for all labor of setting stulls and poling.

Taking up the bottom for track is also variable in cost. A miner using either a Waugh, Jackhamer or Leyner will shoot up from 6 to 10 ft. of bottom per shift. The machine costs, such as power, repairs, etc., will

be noted when breaking ore is considered. Explosive cost is low, as the holes break to good advantage. Track laying is rapid and cheap. Rails and ties from the lift below are usually used. About 10 cts. per foot would cover the labor of removing track from one lift and laying it on a new grade. The total cost per foot for the intermediate track complete is about $1, and amounts to 7 cts. approximately per ton stoped.

These costs mentioned cover preliminary or incidental costs, dead work in a sense. For actual stoping, although costs may vary greatly with conditions in the individual stopes, more constant figures can be given, being obtained from totals of the whole mine over considerable periods of time. About seven tons per drill shift is broken in stopes. The machinemen get $3. Machine costs per drill shift over the whole mine are: Power, 40 cts.; repairs, 40 cts.; lubrication, 2 cts.; and drill-steel sharpening and replacement, 75 cts. The amount of powder used per ton is 1.65 lb., with 5.8 ft. of fuse and 0.9 detonators. A stope hole breaks on an average 1¼ tons. The steel consumption using 12 A Waugh stopers is 2¼ lb. per drill shift.

The average shoveling rate in the stopes is about 7 tons per man-shift. Two rates are paid for shoveling labor, $2.50 and $2.25, and an average would be about $2.35, giving a shoveling cost of 35 cts. per ton. This is the total cost of getting the material into the main-level chutes at the bottom of the stopes, and includes whatever sorting is done underground. The average stoping width of the material sent to the mill is about 4 ft.

Other minor operations are involved in the system of stoping, which add to the cost, such as setting the turnsheet for a new lift, setting the go-devil post and block, etc. It should be noted that the breaking and shoveling costs are referred to the material sent down the go-devil, all of which goes to the mill. A large amount, from 10 to 20 per cent. of the material actually broken, is coarse waste, which must be handled and which is sorted and thrown over the poling in the stope; if included in the tonnage figures, this would boost appreciably the duty per man-shift.

Methods and Costs, Mother Lode Mine, B. C. (By E. Hibbert).—The system of mining at the Mother Lode mine of the British Columbia Copper Co., Ltd., presents some interesting features. The orebody is an altered limestone carrying enough gold, silver and copper to constitute a low-grade ore. It is large, 160 ft. wide on an average, but with a maximum width of 260 ft. The hanging wall is also an altered limestone but carries practically no mineral; from the surface to the 200-ft. level the dip is about 70°.

From the 60-ft. level to the surface most of the ore was removed by the glory-hole system; in places some stoping by the shrinkage system was done but proved dangerous since on blasting large pieces would rip off;

the next system tried consisted of putting in spiral raises, the idea being that solid pillars left on the inside of the spirals would support the workings. In practice each raise, instead of being a screw-thread spiral

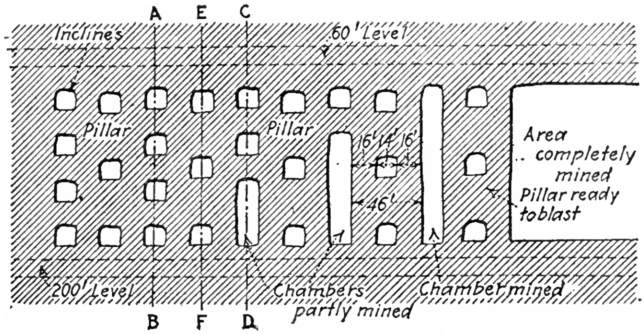


FIG. 154.—IDEAL LONGITUDINAL SECTION OF DEVELOPED OREBODY SHOWING ALTERNATING PILLARS AND CHAMBERS.

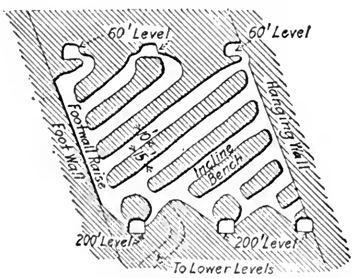


FIG. 155.—FULLY DEVELOPED CHAMBER READY FOR BREAKING OUT BENCHES, SECTION A–B OF FIG. 154.

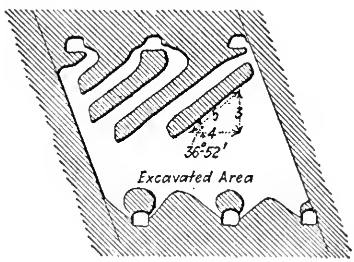


FIG. 156.—CHAMBER PARTLY MINED, SECTION C–D OF FIG. 154.

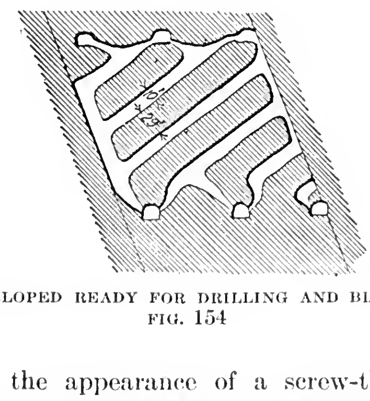


FIG. 157.—PILLAR DEVELOPED READY FOR DRILLING AND BLASTING, SECTION E–F OF FIG. 154

presented somewhat the appearance of a screw-thread stripped from the screw and partly unraveled. Mining difficulties became insuperable and the mine was being cut up with a series of uncoördinated twisted

raises with branches and connections. Furthermore all the underground work was done out of the solid and no advantage was taken of previous workings as free faces for breaking to.

The spiral-raise system was abandoned and by straightening out the raises, and then breaking down the bench between the inclines, making one incline or raise pass vertically over another one, part of the orebody was mined at low cost. This idea was then developed into what can best be described as a pillar-and-chamber system of mining. An ideal, longitudinal, vertical section of the mine between levels with both the chambers and the pillars in various stages of excavation, is shown in Fig. 154.

The preliminary opening up of both pillars and chambers is not dissimilar. A foot-wall raise, shown in Fig. 155, is run, or an old working is used as a raise. This is not straight, but usually a series of spirals, each spiral being made from one incline to that above by partly cutting through the pillars on each side of the incline to make the turn. From this, inclines are driven to the hanging wall at an angle of 36° 52′, the slope of the ordinary 3:4:5 right triangle. Then in the case of the chambers, the benches or ribs between the inclines are broken down, as shown in Fig. 156, a cheap operation, as the ore is bound on three sides only, or, in the case of a first cut on a bench of double thickness, such as those in the pillars, on four sides only. An incline is prepared for benching by mucking off from top to bottom and blowing off the last of the loose dirt with air, in order that a careful search for missed or partially missed holes may be made, partially missed holes being particularly liable to occur when smooth slips are present in the orebody and run with the incline, but at a slightly greater angle than it. Mining the benches is comparatively safe as the men are close to the back and can easily bar down loose ground. Since a man, rolling from the top of a bench, might drop 100 ft. or more, it is necessary to anchor both men and tripods by ropes.

The slope of 36° 52′ has proved the best for getting rid of the broken ore without mucking and still maintaining the incline as flat as possible for ease and cheapness in driving. To ascertain the right slope for any given orebody, will require some experimenting. In driving the incline, if the slope is too great, all the broken ore will run down, leaving the bare rock showing on the bottom, and if the slope is too small, the incline will fill up with the broken ore.

In the pillars, the benches are not broken down, and fewer inclines require to be run, since the thicker benches then left can be drilled out from the inclines above and below the bench, as exhibited in Fig. 157, a transverse section through a pillar. Mining the pillars constitutes one of the interesting points of the system. From the inside, the pillar is drilled out from top to bottom, holes being put in tops, sides and bottoms

of the inclines. The standard length of a hole is 14 ft., although in places where such long steel cannot be changed, shorter holes are put in, and in other cases, where necessary, 16-ft. holes are drilled. The pillars at the end of the mine farthest from the shaft are first drilled and an effort is made always to have several pillars drilled out ready for blasting.

A large porphyry dike, 20 ft. thick, cuts through the orebody at the 200-ft. level, the dip of the dike being toward the foot-wall of the orebody at approximately the same angle as the inclines. The inclines are run above the dike and the pillars blasted from the dike to the surface, the dike being left intact so far as possible, in order that it may form a roof for the workings under the 200-ft. level. In blasting the pillars, 40 per cent. dynamite is used and all the holes in a pillar are loaded and then fired in a mammoth blast of several hundred thousand tons by means of electricity.

The system of mining described here represents the ideal that has been kept in view, but owing to the existence of old workings, it has been considerably modified to suit the conditions. The average costs per ton for the 12 months ending Nov. 30, 1912, using the system described, are given in Tables I and II.

The cost per ton for explosives in a large blast is small, but on account of the old workings it has not been possible to drill out thoroughly all parts of a section, and consequently some of the broken ore from a large blast is caved ground, which has to be bulldozed in the chutes and block-holed and broken up in the inclines leading to the lower levels. It is in this work that most of the explosives are used.

For the successful operation of this method of mining, it is considered necessary to have a large, fairly uniform orebody of too low a grade to permit the use of any filling or timbering method for supporting the excavations made during its extraction. The ore and walls must be good holding ground, and the foot-wall dip at an angle greater than 36°. These requirements limit the application of this system of mining, but as adopted at the Mother Lode mine, because of the fact that the roof is exposed in small areas and the miner is always close to the back, it has proved itself a safe method. The low costs are due to the following causes:

(1) Practical absence of mucking.

(2) Timbering required in chutes only.

(3) Increased drilling capacity per machine when drilling out pillars, as the machine has not to be torn down and stowed away every day for blasting, but keeps drilling all the time; the footage drilled per machine shift, using 3¼-in. machines, has risen from 22 or 23 ft. to 31 or 32 ft. since this system was adopted.

(4) The exposure of free faces by driving inclines which renders the remaining ore easier to break.

TABLE I.—COSTS DISTRIBUTED ACCORDING TO OPERATIONS

Total shipments	377,510 tons
Mining:	Cost per ton
Mining and timbering (labor)	$0.1380
General underground and tramming (labor)	0.0956
Explosives	0.0997
Candles	0.0046
Drill parts and hand tools	0.0210
Shops (except drill parts and hand tools)	0.0063
Compressor and power	0.0269
Lumber	0.0028
Sundries	0.0158
Total	$0.4107
Hoisting:	
Labor	0.0339
Shops and sundries	0.0099
Compressor and power	0.0135
Total	$0.0573
Crushing, conveying and storage:	
Labor	0.0161
Shops and sundries	0.0160
Compressor and power	0.0033
Total	$0.0354
Surface expenses:	
Labor and staff salary	0.0244
Insurance and taxes	0.0227
Shops and sundries	0.0121
Boilers	0.0101
Total	$0.0693
Total, f.o.b. railway ears at mine	$0.5727

TABLE II.—CERTAIN ITEMS SUBDIVIDED INTO LABOR AND SUPPLIES

Operation	Cost per ton		
	Labor	Supplies	Total
Tramming	$0.0556	$0.0071	$0.0627
General underground	0.0400	0.0044	0.0444
Machine shop	0.0124	0.0012	0.0136
Blacksmith shop	0.0154	0.0019	0.0173
Carpenter shop	0.0036	0.0002	0.0038
All shops	0.0314	0.0033	0.0347

Stoping Methods at the Golden Cross Mine (By Andrew W. Newberry).—The country rock, mica-schist, at the Golden Cross mine in southeastern California has a fairly uniform dip, varying from 32° near the

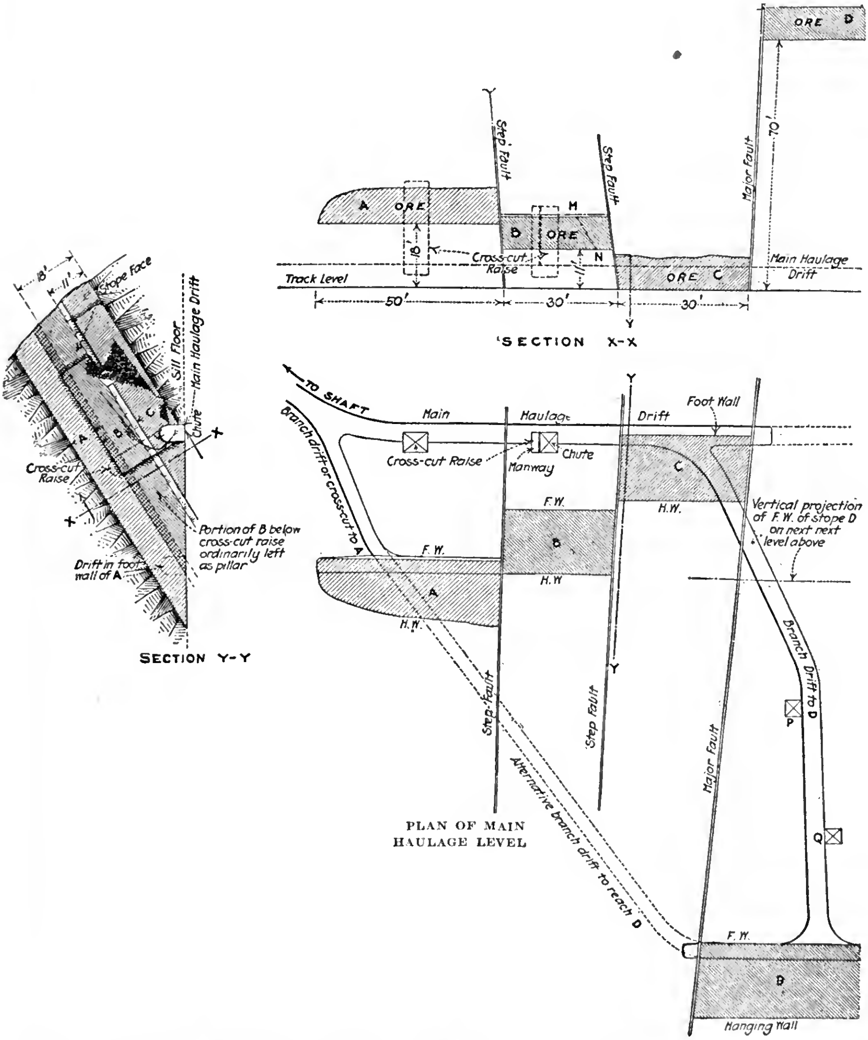


FIG. 158.—PLAN AND SECTIONS, SHOWING RELATIONS OF RAISES, DRIFTS, CROSSCUTS AND STOPES.

surface to 24° at a depth of about 1000 ft. The deposition of the ore followed certain layers which were subsequently faulted so as to give the impression of separate and distinct oreshoots. The principal fault planes are nearly vertical and approximately perpendicular to the strike.

Hence they cut the orebody into blocks fairly continuous on the dip. Step faults, roughly parallel to the major fault, are not uncommon. These show displacements up to 12 ft. with little or no fault breccia, so that their exact position is often difficult to determine until the ore is found on both sides. As the ore is similar in appearance to the altered or silicified country rock, close sampling is necessary. The occurrence of step faults and their relation to the major fault is illustrated in Fig. 158, which gives a section X–X, taken along the strike and perpendicular to the dip of part of the orebody, looking upward on the dip.

Most of the mining is now carried on below the 600-ft. level, where the strata have an average dip of 26°, and an average thickness of 10 ft. and the level interval in the stopes is 113 ft. The method of development on a typical level is shown in Fig. 158. Section C is first developed by drifting along the foot-wall; the foot and hanging of sections A and B are then determined by "crosscut raises" driven perpendicular to the bedding and sampled by means of the cuttings from holes drilled with stopers. Section D, having been proved on the level immediately above, is developed by a branch drift or crosscut from the haulage level in question. This is best driven from C, giving opportunity to raise at P and possibly also at Q and mine the upper part of D through these raises. It would also be possible to crosscut from A and shorten the tram, but this would not permit mining through P and Q.

Section A, which, on the level in question, is of greater importance than B or C, is developed in the same manner as soon as the foot-wall of the orebody has been definitely determined by the crosscut raise. That portion of section B which lies below the crosscut raise, would ordinarily be left as a pillar until A and C are stoped out. The upper part of B is stoped along with A and C, the broken ore going to a chute which is built in the crosscut raise, except that portion to the right of MN, which is handled most economically through the stope C, provided the face of the latter stope is kept somewhat in advance of B, as stoping proceeds up the dip.

In the upper part of the mine where the dip of the formation exceeds 30°, it has been found expedient to connect two levels, through the orebody, by means of at least one 6 × 8-ft. "dip raise," the name locally applied to a raise which follows throughout its extent the dip of the strata. The principal object of such a connection is to afford ventilation. Of the various methods tried for the handling of broken material on a flat dip, the fixed steel chute was found most satisfactory and has been adopted throughout the mine. The dry ore with a considerable proportion of fines is found to slide best on a dip of 32°. At 30°, the fines have a tendency to hang up and block the chute, while at 34° the slide is so rapid that many of the larger pieces jump out. The ideal arrangement for long

chutes was found to be a dip of 34° for the first 10 ft., flattening gradually to 31°, an average of about 32°. This arrangement, for obvious reasons, is only approximated in practice. With dips as flat as 30°, the raise is begun at the foot-wall of the ore on the lower level and terminated close to the hanging wall on the upper level, giving an average inclination of 32° without breaking into the wall rock.

Where the dip of the beds is less than 30°, dip raises from level to level would have added considerably to the cost of mining. The scheme followed is to carry up the stope the full thickness of the ore for a distance of 60 to 70 ft., placing the chutes as shown in Fig. 159, and keeping them up to within shoveling distance of the face. When the stope has been advanced to this point, it is necessary either to install some means of

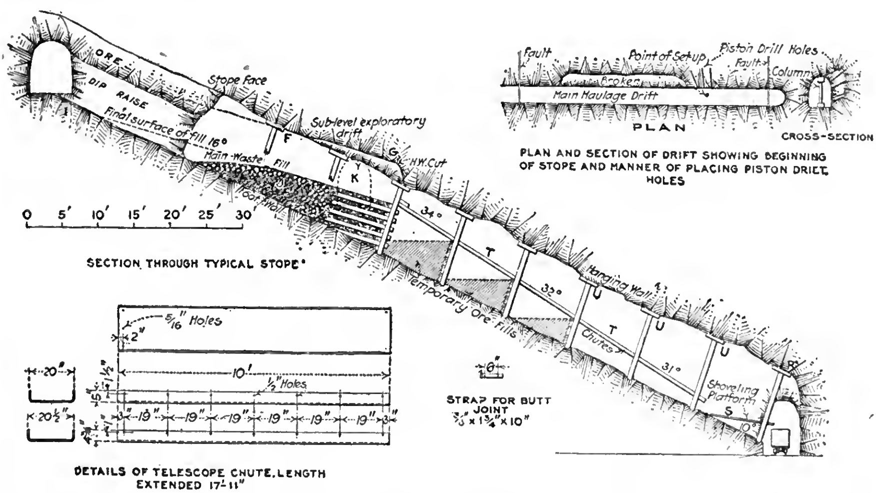


FIG. 159.—BEGINNING OF STOPE, STOPE SECTION AND DETAILS OF STEEL CHUTE.

forced ventilation, which is done in case the orebody is not developed on the level next above; or better, to drive a dip raise along the foot-wall to hole into the drift on the upper level, which is the plan followed wherever possible. Such a raise is kept as small as is consistent with proper breaking, about 4½ × 6½ ft. The broken ore is handled in wheelbarrows which are run up on temporary ore fills and dumped into the chute. As the raise is advanced, successive cuts parallel to *EF*, Fig. 159, are taken off the hanging wall to provide headroom, part of the broken ore being left as a temporary fill. In this way the wheelbarrow runway is flattened somewhat and the handling of the muck from the dip raise is facilitated.

Taking section *C*, of Fig. 158, as an example, stoping is begun at one end and a 4-ft. cut taken off the same height as the drift, using a one-man piston machine, the holes being pointed approximately up the dip. In

Fig. 159 is shown the manner in which this first cut is made. After the first cut, the piston machine is replaced by a stoper and the ore next the hanging broken on 2 × 12-in. boards laid on the track level. The face is then carried upward on the dip full width with a series of cuts similar to the first except that the stoper is used. These latter cuts are somewhat shallower than the first, averaging about 3 ft. 3 in. When the face has been advanced 10 ft. over the full stoping length, a pair of 10 × 10-in. stulls is placed as shown at *R*, Fig. 159. These stulls are set 5 ft. 10 in. center to center and as near the middle of the stope as possible, allowing for the pitch of the oreshoot. Other stulls, usually 8 × 8 in., are placed at intervals along the strike where needed.

When the stope face has been advanced four rounds (about 14 ft.), the bulk of the broken material no longer falls on the track. A steel chute *W*, the details of which are shown in Fig. 159, then serves for loading cars. The lower end of the chute rests on the edge of the car and the upper end is either supported by blocks to give the necessary inclination or suspended from a plug in the hanging by means of a clevis. In this way the trammer can load his own car. After the next cut has been made, the chute *V* is brought in and bolted inside of *W* with four ½-in. bolts. As the face is advanced, the chute is extended until the two parts overlap only 25 in., giving a total length of 17 ft. 11 in. Only one or two telescope chutes are used in a stope as they are readily moved from place to place. To make these chutes, sheets of No. 10 tank steel 30 in. wide by 10 ft. long are bent cold and drilled with 5/16-in. and ½-in. holes, spaced as shown. A chute similar to type *W*, but with the ½-in. holes drilled at each end only, is used for the fixed chutes referred to under dip raises. As only one, or at most two, stopes are brought in at one time, four of the inner chutes *V* answer for the whole mine and the bulk of the sheets are made up in form *W*.

While the broken ore is being loaded by means of telescope chutes, the two 10 × 10-in. stulls *R*, Fig. 159, are tied with a 6 × 8-in. crosspiece and long bolt, and a shoveling platform *S* is constructed of 2-in. planking inclined downward toward the track on an angle of 10°. This is next covered with two sheets of tank steel of the same dimensions as that used for the chutes. As soon as the face has advanced too far to be reached by the telescope chute extended, the latter is done away with, and other pairs of stulls, *U*, are placed roughly in line with *R*, and spaced 9 ft. 9 in. center to center on the dip; 8 × 8-in. timber is used in all stulls except *R*. The pairs of stulls *U* are connected with 4 × 6-in. tiepieces, resting on 2 × 6-in. blocks nailed to the stulls, and turned so that the 6-in. surface in each case carries the two steel chutes *TT*, which are butt-connected at these points; the chutes being set at a steeper angle than the foot-wall, the 9-ft. 9-in. spacing of the stulls accommodates nicely the 10-ft.

chute lengths; 40-penny nails, driven through the $\frac{5}{16}$-in. holes in the ends of each chute, hold the latter in place, and the straps shown are bolted, like fish plates, both inside and outside for greater solidity. A drop of 18 in. is allowed between the lower end of the fixed chute T and the platform S. The two lower chutes are set up at an angle of 31° with the horizontal, the next two at 32°, and the fifth and last at 34°.

As the difference between chute and foot-wall increases, shoveling becomes more difficult. Moreover, when the face has advanced beyond the reach of the telescope chute, a considerable proportion of the broken ore, say 60 per cent. on a stoping length of 35 ft., must be brought to the fixed chute in wheelbarrows. The latter difficulty suggests the installation of shoveling platform and chutes in duplicate, but there is no saving on the stoping length assumed. The plan followed when the chute has reached a height of 4 ft. above the foot-wall, is to lag against the stulls the full length of the stope and allow the broken ore to lie where it falls until a temporary fill is completed as shown. This serves as a wheelbarrow runway for moving the ore from the ends of the stops. When a stope has been finished, these fills are mucked out.

The amount of ore tied up in this way is not large until the stope has been advanced 60 to 70 ft., and at this point the method is varied somewhat. Stoping proper is stopped until the dip raise above mentioned has holed through. As the drills are operated only on the day shift, and the average daily advance is 2.5 ft., this requires about three weeks. Meanwhile, the ore over the raise is slabbed off as described under dip raises, to provide headroom, and a narrow fill is made in the space later occupied by the main waste fill, as shown.

While the dip raise is in progress, some exploratory work is undertaken beyond the lateral stoping limits. This generally takes the form of two sublevel drifts K, one at each end of the stope, driven from a point a little above the middle. Such work furnishes material for the main waste fill which is built up from the ends of the stope toward the dip raise until it encounters the central ore fill. The latter is then mucked out and a cut FGH made in the hanging wall about 3.5 ft. wide to provide headroom for the dumping of wheelbarrows in the upper part of the stope. Holes, 12 to 14 in number, drilled with the stoper, suffice for the hanging-wall cut, and the waste produced fills the space previously occupied by the last temporary ore fill, the width of which is limited by that of the dip raise.

As soon as the dip raise holes into the next level, stoping is begun again, breaking toward the raise. Fig. 159 illustrates this last phase. It shows the stope face advanced about 25 ft. along what was previously dip raise, with 20 ft. still to be broken. The main waste fill is almost completed, its final surface being the dotted line IJ. Waste for this

part of the fill may be obtained most cheaply from development on the upper level. The final surface, inclined at 16° with the horizontal, affords easy wheelbarrow grade from any part of the stope.

The aplite dikes which are often present in the oreshoot constitute a further source of waste. This material, where reasonably hard and unaltered, may be easily sorted from the broken ore, and used for permanent filling in place of the temporary ore fills.

Assuming that 250 ft. of drifting is necessary for the development of the blocks *A*, *B* and *C*, aggregating 10,000 tons of ore of which 90 per cent. is recoverable, the development cost per ton for drifts and crosscuts only, at $4.64 per foot, comes to 15 cts. Adding to this 3 cts. per ton as the cost of crosscut raises and sublevel exploration work, 18 cts. per ton is obtained as a fair estimate of development costs on a block of this size.

Where the dip raise is not driven until the stope has been advanced over half the level interval, its cost properly falls under the head of extraction. In case the dip raise is driven through from one level to the next before stoping is begun, as is done in the more steeply dipping portions of the mine, its cost belongs to development.

An average cost per foot for 100-ft. dip raises is about $5.40. Adding the cost of two 100-ft. dip raises to the development previously assumed increases the development cost on the 9000 tons recoverable by 12 cts. per ton, giving a total of 30 cts. The dip-raise cost is entirely offset on dips greater than 30°, by saving in extraction, since one round out of each cut in stoping is in effect a dip-raise round, and the work is carried on under better conditions.

Allowing for the initial round of each cut or lift in stope advance, the average break per machine shift is a piece of ground 3 ft. 3 in. by 7 ft. by 10 ft., or 17.5 tons. An average round is nine 3½-ft. holes with 30 sticks of powder. The average cost of extraction per ton for the typical stope is about $1.07. distributed as follows:

Stoping Costs per Ton

Labor, breaking	$0.18	Chutes	$0.01
Labor, shoveling at face	0.26	Pipe and fittings	0.00
Labor, loading cars and tramming	0.14	Candles and lubricating oil	0.03
Labor, timbering	0.07	Power and water	0.07
Labor, blacksmith	0.04	Drill repairs	0.01
Explosive	0.14	Blacksmith coal and drill steel	0.01
Timber	0.09	Superintendence	0.02

These costs cover breaking and tramming only. No account is taken of hoisting, sampling, assaying, office expense or depreciation. The item of superintendence covers only the foreman's wages.

Mining a Pillar of Magnetite.—In mining the deposits of magnetite at Mineville, N. Y., the room-and-pillar method is adopted. In an exceptional orebody these pillars may be trimmed down to a floor of over 100 ft. below the back. Often in taking up lower levels, the pillars which are started on the floor above show signs of weakness, due to seams and blocky ground, when rounded on the lower levels. Trimming the pillar or cutting away portions to seams may make the pillar perfectly safe. Again a seam may show which cuts across the entire pillar and at such an angle that blasting near the ground will dislodge portions above the seam.

A particularly seamy pillar in one of the Mineville mines was taken down, after it was decided that the danger to the miners working near such a pillar would be much greater than working under the greater hanging-wall span left after removing the pillar. To mine the pillar, reliable men were selected, A-frame staging and ladders were rigged, and two roofmen (working as machinemen) cut the ore away in slices until the pillar was lowered below the main slip. The work was supervised by a head roofman. Two miners were kept near the foot of the pillar under cover of the back of the main haulage drift. They hoisted drills and water and tended the guy and pulling-back ropes. The miners on the pillar worked with ropes around their waists. These ropes passed through small sheave wheels held by split eye-bolts wedged in holes in the hanging wall. Several sheaves, each fitted with a 5/8-in. rope, were hung, and new eye-bolts were fastened to the hanging close to the breast as the cutaway advanced. If any fall from the back had broken the staging, the men could have been lowered by the miner at the foot of the pillar, by means of any one of the ropes hanging from the sheaves hooked to the eye-bolt.

The ropes served their purpose, for after one of the first blasts near the hanging wall, the head roofman had climbed the staging to inspect and had started down the back ladder when he heard the back cracking. He climbed back to the small bench, which had just been made by blasting. The fall broke the A-framé and ladders and he was lowered by one of the ropes.

The first drilling on the pillar was done with a block-hole machine; after a small bench had been cut in near the hanging wall, a 2 3/4-in. piston machine was used. Blasting was done by both fuse and battery and holes were pointed in such a manner that little material went in the direction of the staging. Good judgment was exercised by the miners in blasting, for until the pillar was entirely free from the hanging, the roofmen left the complete drilling outfit on top of the pillar during each blast and the machine was never broken, nor was any part of the staging broken by blasting, except a few ladder rungs. Before spitting the holes, the rigging was pulled back some distance by block and tackle. A

small portion of the pillar came away on the line of the main slip during blasting This was fortunate, for it made the work much safer. The one block which did slip off weighed approximately 170 tons. Fig. 160 shows approximately the order of blocks removed in freeing the pillar from the back, and also the staging and equipment.

In raising the staging, the following was the procedure. A 30-ft. ladder was raised and an eye-bolt fixed at A; 6 × 6-in. stringers, spliced

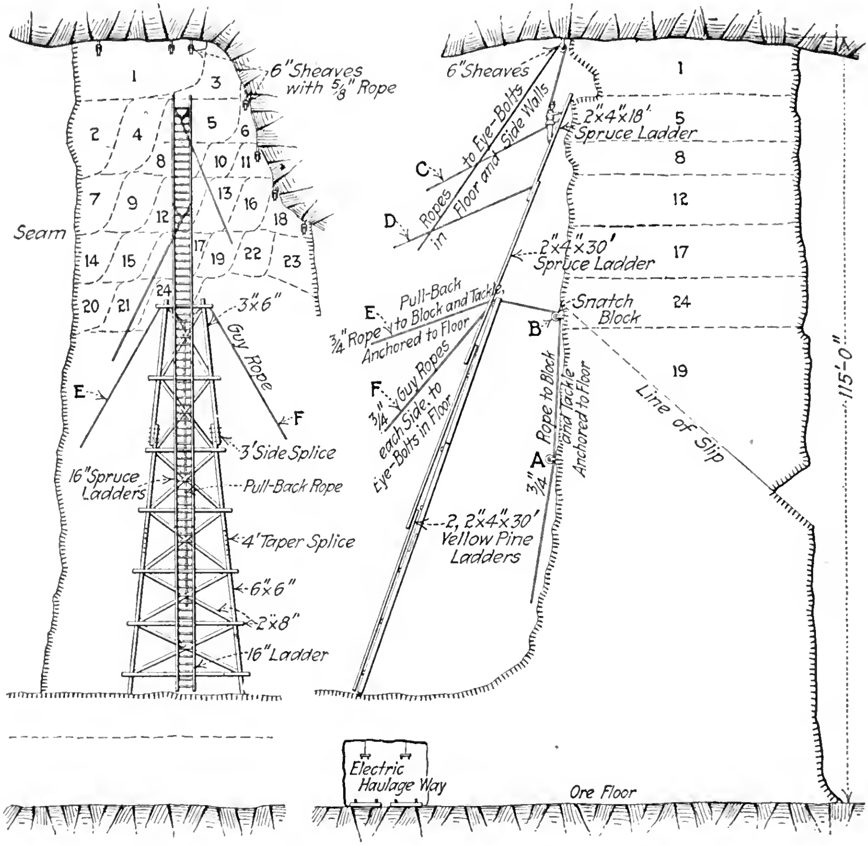


FIG. 160.—METHOD OF SLICING A PILLAR OF IRON ORE, MINEVILLE, N. Y.

by 2 × 8-in. plank, were then raised by block and tackle hung to the eye-bolt. Spliced spruce ladders were then raised on the staging to a point B, where another eye-bolt was driven. By means of this and block and tackle, 3 × 6-in. sticks were then raised, spliced to 6 × 6-in. stringers, and then strutted with 2 × 8-in. plank. From there on ladders were spliced to the bottom length of ladders, and the line was drawn up to the top of the pillar by block and tackle hitched to the 2 × 8-in. braces,

the ladders being guyed by ropes *C* and *D*, and the staging guyed by ropes *E* and *F*.

Top-set Slicing on the Mesabi Range (By L. D. Davenport).—In one of the small mines of the Chisholm district, in Minnesota, top-set slicing was tried. The ore in the working place in question extended about 21 to 23 ft. above the level and to take it out, the miners proceeded as follows: First an open set was put in the drift using 7-ft. caps and 12-ft. posts (sets 1 and 2 shown in the plan Fig. 161). Next, the slice was extended along the caved ground putting in 7 × 12-ft. drift sets spaced about 5 ft. apart for about 25 ft., which was the limiting distance on that side. Then the ore above the last sets, 6 and 7, was taken out and 7 × 8-ft. sets were put on top of the 7 × 12-ft. sets already in place. This top tier was then worked back to the open set. The caps of the bottom sets were framed on top to make a bearing for the posts of the top sets. Seven sprags were used between posts and caps of the

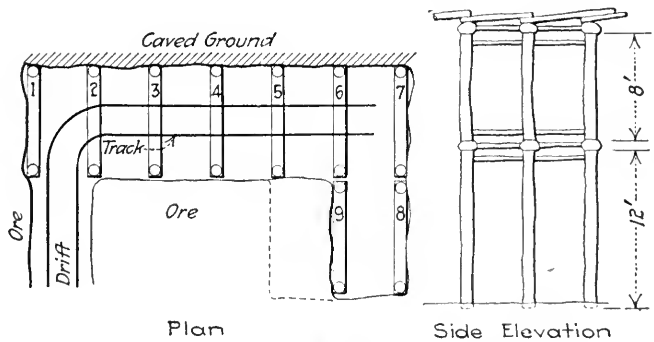


FIG. 161.—UNSATISFACTORY METHOD OF TOP-SET SLICING.

top sets as shown in the elevation. The next step was to make room for sets designated in the plan as 8 and 9; this time, the full height of the ore was taken out and both top and bottom sets were put in. In taking out the ground adjacent to 9, for the next set, the side weight of the unsupported caved ground caused set 5 to move and although the miners tried to hold it with props, they could not stop its moving and the set "jacked-knifed," the end of the room filling with sand.

In another similar room the plan was tried of taking out the top set adjacent to set 9 first, then putting in the cap of the bottom set as a sill for the top set and then coming in under it with the 12-ft. posts. This method did not seem to help much, for as soon as the second slice was opened up the whole room started to move. As soon as the posts moved an inch or so, one end of the sprags would drop down and then a little working in the caved ground would throw down a set. In a single slice the top sets worked well but in widening out for the second slice,

as soon as the ground was relieved at the side of a set, the timber started to move. It is usually difficult to blast down a single top-set slice, for even when all the bottom posts are blasted the caps act as stulls and hold up the top sets.

It is claimed for this system that less timber is used and that the miners have less shoveling to do, on account of running the ore from top sets directly into the car, than in taking the same ground in two slices.

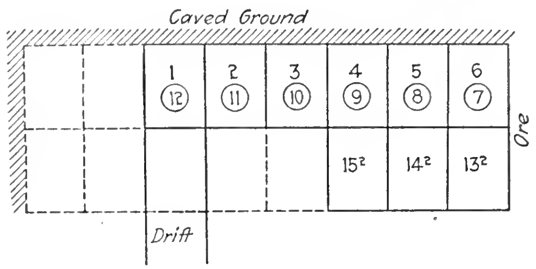


FIG. 162.—SEQUENCE OF SET REMOVAL SHOWN UNSATISFACTORY.

It was found that the amount of timber used was about the same with the exception of the boards used on the bottom of the upper slice. The miners stated that the time saved in shoveling was more than offset by tightening and replacing sprags, putting in props, etc. But the most important thing is the question of the safety of the men. A regular 10-ft. slice is certainly safer to work in than a 21-ft. top-set slice. In this particular case, the miners put up two raises in the drift

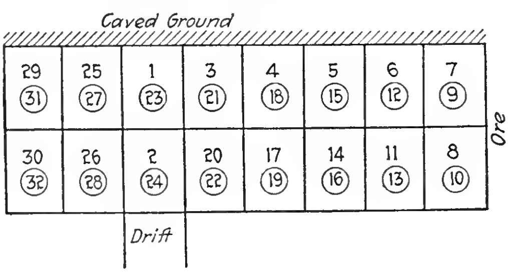


FIG. 163.—POSSIBLE IMPROVEMENT IN SEQUENCE.

and took out the remainder of the pillar in two slices. They did not have to hoist the timber any higher than in the top sets and the slight additional cost per cubic foot of the top drift and the raises, as compared with the same amount of ore mined by slicing, was negligible. Also, the miners get out a few tons per day more than with the top-set slicing.

Fig. 162 is a plan showing the order in which the sets were taken out in the working place described. The plain figures refer to bottom

sets, those in rings, to top sets; and those with superscript, to both. Fig. 163 shows a proposed method of taking out the ground with top sets, which might work better, although it has not been tried here. After putting in the open sets 1 and 2, the bottom sets are taken out along the caved ground to the limiting distance, say to set 7; then, instead of raising up and taking out the top sets, set 8 is cut in for on the bottom while 18 or 9 ft. of solid ground above holds back the caves. Next, set 9 on top of set 7 is taken and then set 10 on top of set 8. The remaining sets are taken out in the order indicated. Another proposed method, which is similar to that used in square-set work in heavy ground, is shown in Fig. 164. The open sets 1 and 2 are put in the drift and the top set 3 over 2 is taken out; then set 4 in the solid is taken and the top set above it, etc., taking the ground in vertical tiers. As soon as set 13 is finished, the solid side of the room is boarded up. Next, set 14 is taken out on the bottom while the ground above it steadies the caves, then set 15 is taken and at 16 another bottom set started.

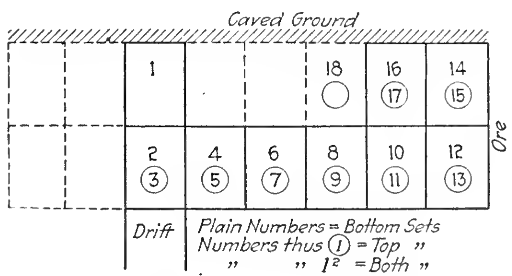


FIG. 164.—PROPOSED SEQUENCE.

Successful Top-set Slicing (By Pomeroy C. Merrill).—A method of top-set slicing which has been used successfully for some years in one of the larger underground mines of the Mesabi range is illustrated in the accompanying diagram. It has been used in hard ore, in clayey, wet, crushing ore and in soft hematite, under caved ground and under sand.

The numbers in circles in Fig. 165 refer to the top sets when the two-set method is used. In such case, the order represented by the figures in the drawing is followed; the cut is taken in for 35 ft. on the far side of the room next to the cave, removing lower sets only, and then four lower sets returning on the near side are removed and top sets are mined in the order shown. The room is then carried back toward the open sets under 50 and 51, the miners keeping the bottom set of the returning side ahead of the top sets on both sides, but sets 52, 53 and 54 are left standing, and the room is squared up, boarded and the timber blasted out. The smaller room on the other side of the drift is then mined similarly, in the order shown, and the timber there also blasted out.

In blasting down the mined-out rooms, it is seldom necessary to blast more than the center posts of the top set. In this way the room fills before the boards on the sides have had a chance to move. It is no rare sight to see every cap in the bottom sets of the far slice next to the caved ground resting on a post originally placed in the near slice. Blasting is done by battery, and, together with the boarding up, is conducted by two blasters who go from room to room as required.

The advantages of this method are that it saves timber, since only one post is used in the center and frequently old posts are used on the far side; it saves shoveling, the top-set ore being run into cars through board chutes which can be quickly placed in any set; and it is safe.

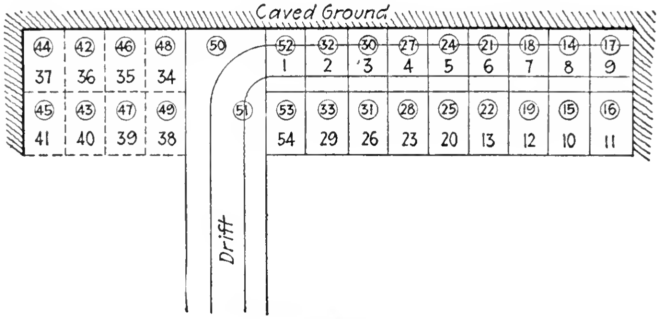


FIG. 165.—TOP-SET SLICING METHOD PROVED SUCCESSFUL.

Removing Ore under Timbered Drift (By H. H. Hodgkinson).—Sometimes it is necessary to remove ore from under drift sets which have been loaded with rock-filling on both top and sides, and to do this without disturbing the set timbers, so that the drift is still maintained. The following method has been used successfully and economically, not only keeping the set timbers and filling intact, but permitting a complete recovery of the ore.

On the level below, set timbers with chutes were installed and two raises, Fig. 166, driven to the level above at each end of the orebody, so as to afford an entrance and exit to the working place after the chutes in the sets were full of broken ore. The ore was then stoped to within 12 ft. of the level above. The stope was drawn and filled with waste up to within 5 ft. of the back, a crib chute being built as the filling progressed, for handling the ore which would be obtained by mining the 12-ft. floor pillar. The fill was placed in the empty stope by means of the two raises until it reached its angle of repose, when it was leveled off to an elevation of 5 ft. below the floor pillar. The remainder of the fill was placed in the stope by means of a small mine car which was lowered down one of the raises and supplied with fill by a chute placed at a raise bottom.

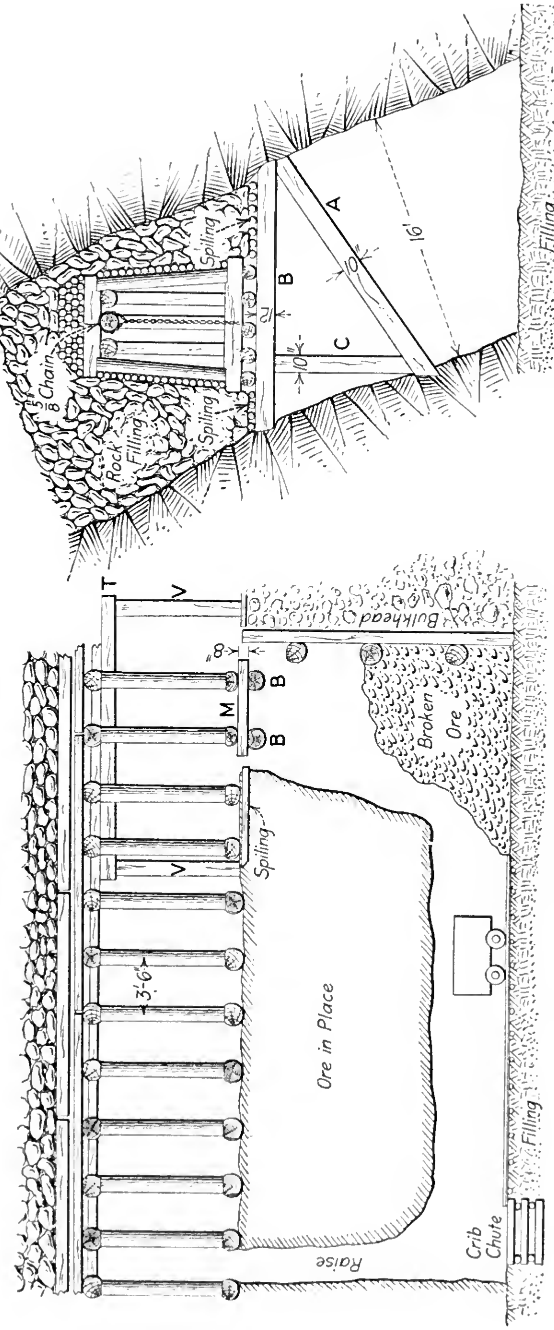


FIG. 166.—SECTIONS SHOWING TIMBERING TO CATCH UP DRIFT SETS

Before mining the floor pillar the three 12-in. oak timbers *T*, 15 ft. long, were placed under the caps, being supported by 10-in. posts, *V*, as shown. Each of the set sills was then chained to the caps by means of ⅝-in. iron chains made tight so as to hold the legs and sills of the sets in place while the ore was being removed below. The fill on the sides of the set timbers fortunately was coarse rock; it was spiled off with lagging each time a sill was exposed and was thus prevented from running. The ore was mined from beneath two sets, these being immediately supported by the timber trusses *A B C*; then the ore was mined from beneath the third set, which was supported in the same manner. These timber trusses *A B C* were erected as soon as possible after the sills were exposed, the timbers *A* and *B* being placed in hitches in the foot-wall. The timbers *B* were about 8 or 10 in. below the sills, to facilitate placing them in position, the space between them and the sills being blocked up tight by means of the timbers *M*. As a precaution two props were placed under the timbers *M* to relieve the timbers *T* of some of their weight while the trusses were being framed. The timbers *M* were placed far enough apart to permit similar timbers used under the next sill to interlock with them. The broken ore was not removed until the exposed sills had been supported, and these made a handy stage from which to work. After three timber sets had been supported firmly by means of the trusses, the timbers *T* and *V* and the chains were moved along and used to support three more sets, the ore from beneath these being removed as before. When the ore had been removed from below five timber sets a bulkhead was built as shown and the space back of it filled with waste.

TIMBERING AND SUBSTITUTES

Square-set Timbers for Soft Iron Ores (By L. D. Davenport).—The framing of square-set timber in the Chisholm district of the Mesabi range in Minnesota, is done by contract. Sixteen-foot round timbers, about 12 to 14 in. in diameter, are used for posts. These 16-ft. pieces are cut in two and framed on one end for bottom posts and on both ends for top posts. The tenons are 4 in. square and 4 in. long, thus making the bottom posts 7 ft. 8 in. from end to shoulder and the top posts 7 ft. 4 in. between shoulders. The caps are cut from 14-ft. round timber, about 10 to 12 in. in diameter. The ends are flattened to 9 in. in thickness to make a bearing for the shoulders of the posts. A 4 × 9-in. face is left on each end of the cap to fit against the tenons of the posts and the remainder of the end is cut off at 45°. The caps are 9 in. thick and the length of the corresponding tenons on the posts is 4 in. for each post, which leaves 1 in. between the ends of the tenons when the timber is first put in place. This 1-in. space is left to allow the caps to crush down

slightly before the tenons bear on one another. This cushioning effect of the caps prevents the post from splitting when the room takes weight. As may be inferred from the style of framing, the most of the weight in the stopes comes from the top. In Fig. 167 are shown the templates, which, with an ax, adze and crosscut saw, constitute the outfit of the timber framer.

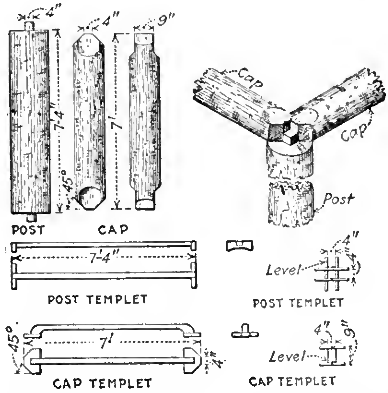


FIG. 167.—ROUND SQUARE-SET TIMBERS AND TEMPLATES.

Square-set Framing at Butte (*Bulletin,* American Institute of Mining Engineers).—In the earlier operations in Butte, Mont., timber was abundant for mining purposes, and it was common practice to use sawed lumber for the stope square-setting, the framing on this being relatively simple. There were, nevertheless, various methods employed for framing these square sets. In 1, Fig. 168, is shown the framing used in the High Ore mine; the post has a long and relatively slender horn, subject to breakage under heavy side pressure, and to crushing with heavy top pressure. The opposite extreme is shown in 2, the framing used at the Syndicate group. In this case the horn of the post is only 1 in. high, which gives a small shoulder for the cap and girt. Furthermore, the framing is unnecessarily complicated, and consequently expensive. In 3 the method for the Anaconda group is exhibited. It is probably in general the best method for sawed timber, being cheap, simple, and designed to obtain the full strength of the timbers in all directions. The horns on the posts, being 6 in. square by 2 in., are strong, and at the same time give a good shoulder for the cap and the girt. The caps butt end to end. The girts in this figure are unnecessarily large, inasmuch as they lie parallel with the strike of the deposits and consequently take less weight than the caps, which are subject to the crushing force of the hanging wall. The framing shown in 4 and 5 is similar to this, except that the girts are 4 in. less in lateral thickness,

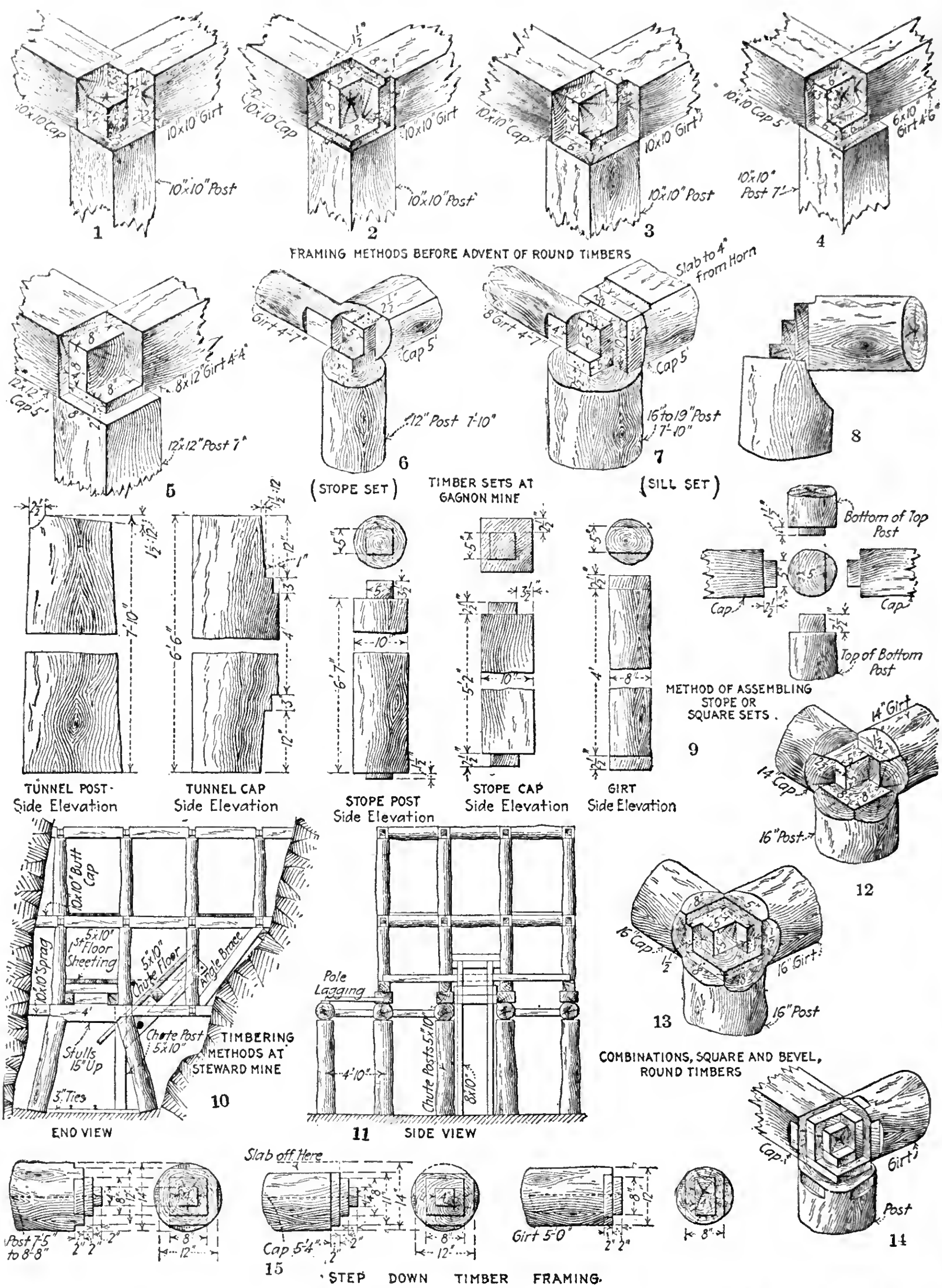


FIG. 168.—METHODS OF FRAMING AND ASSEMBLING SQUARE AND ROUND TIMBERS IN BUTTE MINES.

and so require no framing at all, although leaving the set just as strong. One case shows the dimensions for 10-in. caps and posts, and the other for 12-in. Using the 10-in. timber the sets are made 5 ft. from center to center, either cap-way or girtway, and 7 ft. 6 in. center to center in height.

As the timber about Butte available for sawing became scarcer, it was found necessary to adopt round timbers underground. Even before this, round timber had been used by the Gagnon mine. The method of framing employed here is that shown in 6. The posts and caps have horns 5 in. square by 2½ in., and the cap has a shoulder taken off the bottom 5 in. from the center to allow it to fit snug on top of the post, while on the top, also 5 in. from the center, a slab is taken off for its full length, in order that a stope floor may be easily laid. The girt is framed as shown, being usually less than 10 in. in diameter. The sets are 8 ft. 3 in. from center to center in height, and are 5 ft. center to center in the other two dimensions. The framing used on the levels, shown in 7 and 8, is similar, except that the posts are given a batter of 6 in., furnishing additional resistance against side pressure.

A combination of round and square timber has been used at the Steward mine for a number of years. The caps are of square timber; the girts and posts are round. The dimensions and method of framing are shown in 9, while 10 and 11 exhibit the relations of the level timbering and the stope set. The horn on the post is 1½ in. deep on the bottom, and 3½ in. on the top, it being supposed easier to stand a post with a short horn when erecting the set. This necessitates that the horn of the cap be out of center; this does not weaken the cap, but the timbers, being unsymmetrical, are a little slower to handle. The sets are 7 ft. center to center in height, 5 ft. 2 in. center to center along the cap, and 4 ft. 10 in. center to center along the girt. The drift posts are framed on the top only, and are given a heavy batter, which has proved satisfactory. In 10 is also shown the method of putting in the sheeting on which the waste is filled. The height of this above the drift cap allows room for repairs when necessary. The material used for this is usually small round timber or large round timber sawed in half.

When round timber began to be more generally adopted, the first framing, done in machines, was that shown in 12, the post having a flat top, 8 in. square, with no horn, and with a miter cut to the outside of the timber. The caps and girts both had horns, as shown. While this style was used for some time, it was found difficult to set the timbers and to block them so that the sets would satisfactorily resist pressure, and the concussion of blasting. The first improvement was that shown in 13, a combination of the square and the beveled type. As can be seen, the framing of these caps is complicated, there being a 1½-in.

shoulder at the base of the horn, from which the miter begins. The sets measure in height 7 ft. 5 in., and 5 ft. 10 in. either cap-way or girt-way, in these latter two cases. A still further improvement is that shown in 14 and 15, the step-down type, the large 12-in. horn on the post being used when the post is large enough. The cap is slabbed off on the top in order to enable a level floor to be laid. [The cap in the assembled view, 14, is incorrectly shown as a square timber.—Editor.] In this style, the size of the set was also changed, it being made 7 ft. 9 in. in height and 5 ft. 4 in. in each of the other dimensions; being thus smaller along the caps and girts, it is stronger, and the common length of lagging used, 16 ft., will cut into three pieces to fit the sets and leave no waste. It is probably the best method of framing round timbers where the size varies, the full strength of each member of the set being obtained regardless of the size of the timber. If the timber were all the same diameter, a simpler method could be used, somewhat similar to that shown in 3 and 4 for square timber.

On the main levels, specially selected timber is used, varying from 12 to 18 in. in diameter for posts and caps. There is no difference in the method of framing except that the posts are framed on one end only. It is the intention ultimately to apply this method in all of the mines of the Anaconda Copper Mining Company.

Square-set Timbering at Switches (By Frederick W. Foote).—At the Butte & Superior mine, Butte, Mont., the haulage tracks in places pass through large stopes which require special care in timbering. The insertion of switches and curves was accomplished, as shown in Fig. 169, without disturbing the general system of timbering and without bringing undue strain on any of the parts. At the point where the switch or curve was to be placed, a post of the sill floor, and the corresponding post of the floor above were left out. Where the top of the upper post would have come, a block was placed. Four diagonals reached from this block to the tops of the sill sets. In this way the post of the floor above was supported.

Supporting Back While Drawing Shrinkage Stope (By H. H. Hodgkinson).—Where the shrinkage system is used for stoping and the ore is later drawn through chutes below, it is often found necessary to timber the back after breaking has ceased. In order to obtain the maximum strength and protection from the least amount of timbering it is necessary to frame the timbers in such a manner that each set conforms as nearly as possible to the shape of the ground and is placed as close to the back as conditions will permit. A space of at least 10 in. must always be left between the ground and these sets in order to place the lagging and the blocking; but a greater space only adds to the expense in requiring extra blocking, for as a general rule this space must be blocked up tight to

hold loose ground in place. If the ore is mined in such a way that the back forms an arch, there will be much less weight upon the timbers to support the ground and the sets can be placed farther apart.

In putting up a set, the main timber is placed in two hitches, if horizontal, and in one hitch with a good head-board at the upper end, if it is a stull. It is always best to cut the timbers to fit the hitches and

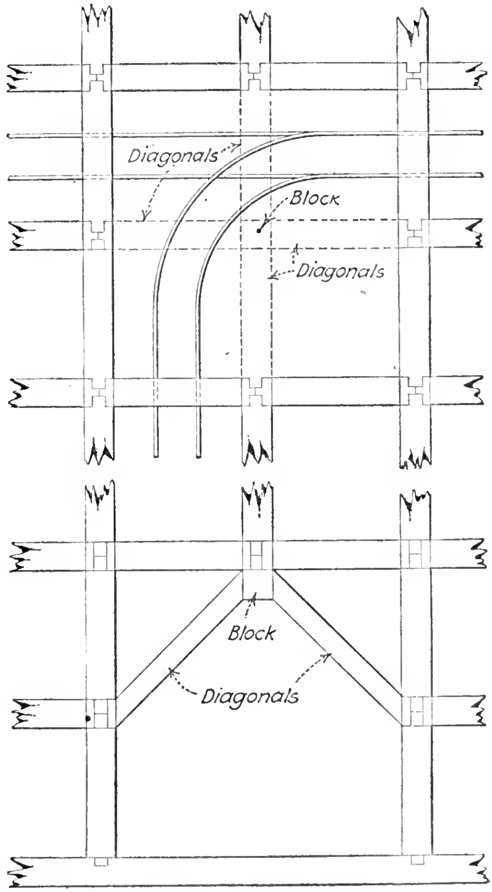


FIG. 169.—OBTAINING ROOM FOR TRACK CURVES IN SQUARE-SET STOPE.

face a timber off at the point where another timber rests upon it, for the more perfectly the joints and bevels fit, the stronger is the set. Considerable time and labor will be saved in cutting the miter of two timbers by bisecting the angle formed by them and cutting each timber at a bevel of that angle. This will insure a much better fit and a stronger job. Both time and labor can be wasted by cutting the bevel on one timber and then cutting the bevel on the other to fit. When the bevels

on the timbers are cut by the hit-and-miss method each timber may not carry its due portion of the load at the miter and the set is liable to fail under stress.

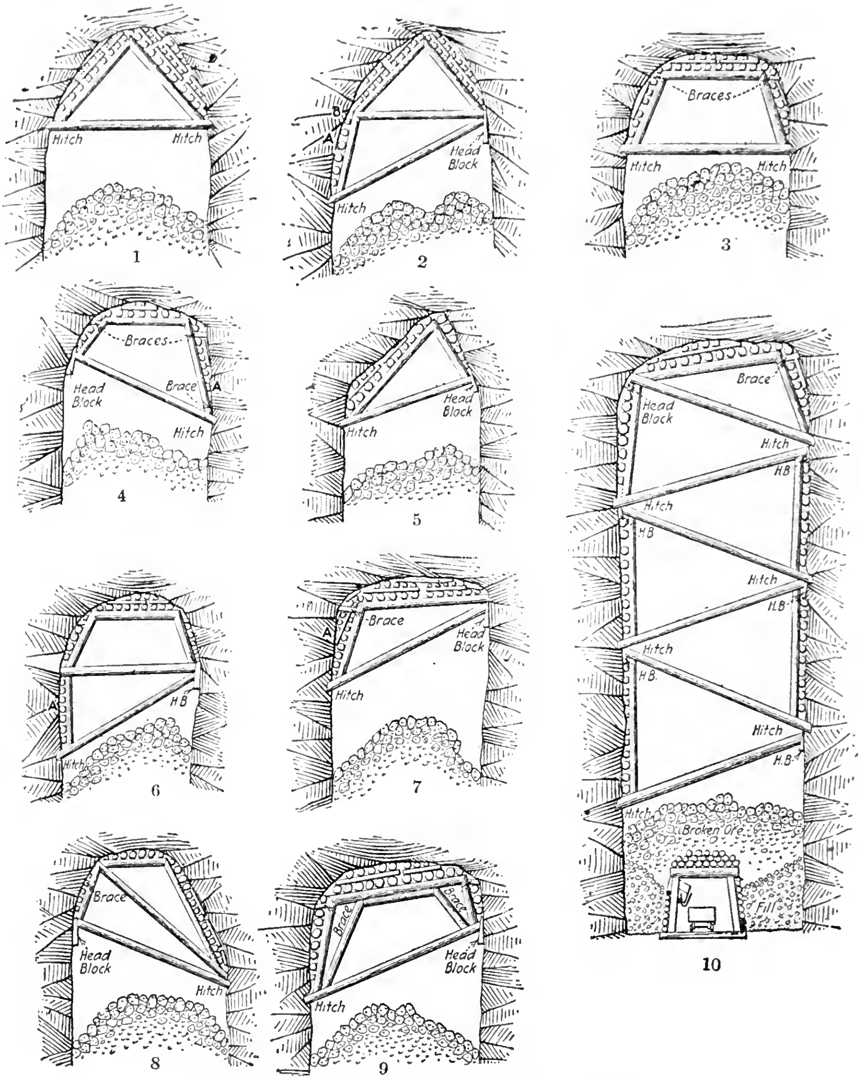


FIG. 170.—TIMBERING FOR CATCHING UP STOPE BACKS.

The sets are placed on from 4½- to 8-ft. centers, depending on the conditions, and are braced against the ground. To keep the miters from pulling apart, they are also spiked with 8-in. spikes.

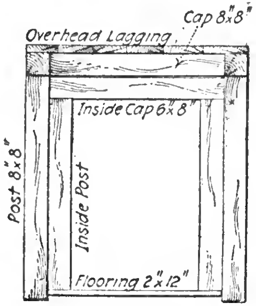
The style shown at 1, Fig. 170, is the most desirable, because of the

highly arched back which permits placing the sets at a greater distance apart. Three timbers are used to make the set, which conforms nicely to the shape of the roof and requires a small amount of blocking. The horizontal timber is placed in a hitch at each end. At 2 is shown a back highly arched as in the preceding case, but with the side *A* in bad ground. Only one hitch is required in this instance, with a good head-board at the upper end of the stull. At 3 is shown a stope whose back is only slightly arched. Two hitches are required for the timbering. At 4 the back is only slightly arched, and the wall at *A* is assumed to be bad and needs more timbering, although only one hitch is used. The back may be highly arched, but due to irregularities in the ground the peak of the arch is not in the center of the stope. The style of timbering shown at 5 is applicable. At 6 is shown a stope with a back only slightly arched and a loose side at *A*. The style of timbering at 7 is adapted to a stope with a very flat back and a bad side at *A*. Timbers placed as shown at 8 not only conform to the shape of the ground, but will support heavy ground. The style shown at 9 will support excessively heavy ground and is so framed that the miters bear their proper share of the burden. As a rule, if the miters and hitches are properly cut, timbers for this kind of work need little bracing. It is poor practice to brace one timber so that the weight is concentrated on another timber, unless the second timber can be directly supported at the point where the load is concentrated.

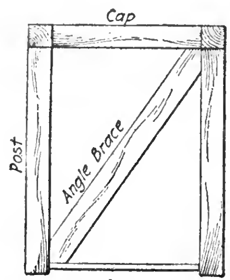
The sides of a stope may be so bad as to require timbering almost all the way down to the floor below. The style used in 10 will accomplish this effectively, each successive stull and lagging with upright piece being placed as the broken ore is drawn down. Each stull is set in a hitch and has a head-board at the upper end.

Top-slice Timbering at Bingham (By D. W. Jessup).—The timbers used in the system of top-slicing practised in Bingham Cañon, Utah, are framed similar to those used in the square-set system; their characteristics are shown in Fig. 171. With the adjustment and settling of the overburden the pressure on the timbers may become enormous, and as it is necessary to keep various runways and connections open, the timbers supporting the runways must be reinforced. This is accomplished by doubling-up sets, helping sets, angle bracing, stulls, a tee piece, etc.

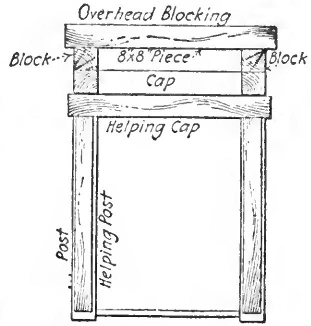
The doubling-up set, Fig. 172, is a complete set placed inside the weakened timbers, and consists of a cap and two posts. The caps are 6 × 8 in. and either square or round timbers are used as posts. The helping set, Figs. 174 and 177, is similar to the doubling-up set and is placed outside the failing set of timbers instead of underneath. This set is generally used for strengthening, as the weight falls more upon the helping post and less upon the cap, causing a more nearly equal distribu-



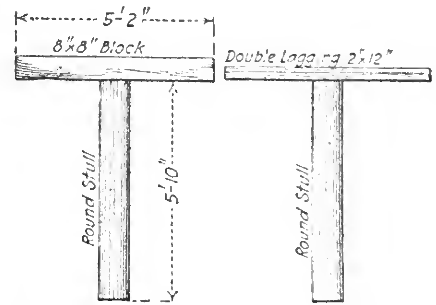
DOUBLING-UP SET
FIG. 172.



ANGLE BRACE
FIG. 173.



HELPING SET
FIG. 174.



TEE-PIECE FOR SPILING
TEE PIECE
FIG. 175.

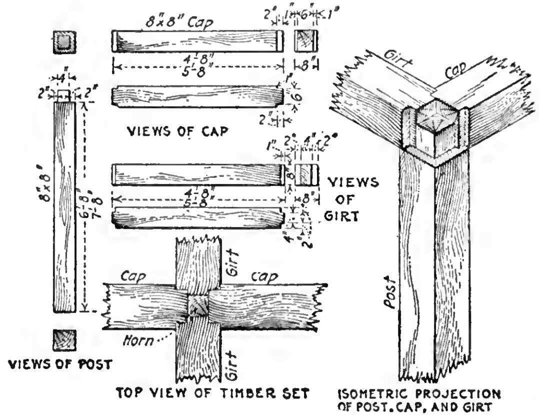


FIG. 171.—DETAILS OF SQUARE SET FRAMING.

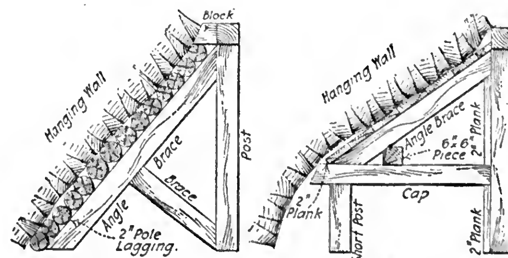


FIG. 176.—ANGLE SETS FOR SUPPORTING HANGING WALL.

tion of weight. The cap in the doubling-up set is often crushed and requires an extra post in the center which may block the passageway. The angle brace is used to withstand the lateral pressure. As the stope is opened the hanging wall begins to slack and swell, causing the timbers to ride or move in a horizontal direction and making the set diamond shaped. This tendency is partially overcome by the angle brace, Fig. 173. The brace is cut so that the ends fit snug in the corner formed by the post and cap. When the cap or girt is breaking under pressure a stull is used for a temporary support, but if it is to remain any length of time the stull will probably buckle and the weight had best be taken up by a helping set. In place of a stull, a tee piece, Fig. 175, consisting of stull and a

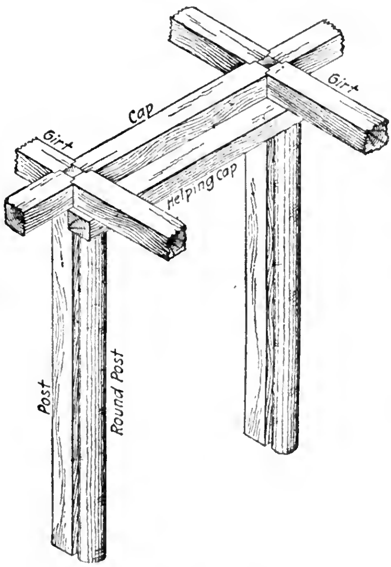


FIG. 177.—ISOMETRIC PROJECTION TIMBER SET AND HELPING SET.

block or head-board is often used. It is also used to advantage when the lagging from the floor above tends to drop and cause a run or deposition of the overburden. The head-board in this case consists of two pieces of lagging. Fig. 176 illustrates angle sets used in timbering a heavy hanging wall. Often there are odd corners, narrow widths and stringers of ore that are mined from one of the floors and may be three or four sets wide. Instead of using the regulation timber set, a combination of tee pieces is sometimes employed and gives good results, especially if that section of the stope is not to remain open for any length of time. The timbers are not placed in any order or system but under the weakest points of the floor above. By this method a great deal of timber and labor is saved.

As each successive slice is mined, the overburden becomes more nearly self-supporting and the mat of timbers forms a more compact mass that assists in supporting the overburden and thus causes less weight to fall upon the timbers. This offers the opportunity for use of the tee-piece system of timbering, and in the future one of the large orebodies at Bingham may be mined by this method instead of the slice-set system. But in most of the orebodies it would not be applicable, as the overburden is in a fine state and is constantly running down from floor to floor, requiring closely laid flooring and substantial timbering to hold it in check.

One of the advantages of the top-slice system is that the doubling-up and reinforcing timbers may be of a cheap material. They are not permanent, being in place merely for a few days, perhaps weeks, and odd lengths, sizes and nondescript timbers can be used. A large amount of the round timber is quaking asp. It is a cheaper grade than pine or fir, possesses less compressive strength and does not withstand rot as well, but it gives satisfaction. Nothing smaller than a 6-in. end is used and the bark is peeled to prevent rotting. For flooring, odds and ends may be used, since the purpose is to prevent the overburden from running through into the slice below, though a well-laid flooring offers better facilities for shoveling. The top of the slice may or may not need lagging, it depends on the condition of the flooring from the slice above, the latter is usually out of place and is not in position to hold the overburden. The broken timbers in the mat are constantly moving downward and exert a great pressure on the timbers below, causing them to break. These projecting ends are sometimes sawed or cut; if over a post they are blocked against it, but never under any circumstances are they blocked against a cap or girt as the pressure would cause them to break. In blocking the timbers, the wedges are always driven under the bottom and not over the top. When the slice has been finished, a number of unbroken timbers are removed and used again in the following slices.

All of the timbers are framed to dimensions in the company's sawmill. No framing is done underground other than an occasional butt cap or short set. The round timbers are from 10 to 20 ft. long and are cut to the required length by the timberman. The following are the different-sized timbers used in one of the stopes: 8 × 8 in., 39 per cent., for posts, caps and girts; 6 × 8 in., 5 per cent., for doubling-up sets and stringers; 2 × 12 in., 38 per cent., for flooring, side and top lagging; round, 18 per cent., for doubling-up posts, stulls and cribs.

Hook and Staple for Staging.—In topslicing on the Mesabi, the face of the ore in a room may be as much as 14 ft. high. To reach the top of this for drilling, a staging is necessary. The typical set of timber consists of two posts and a cap, with no studdle the long way of the room.

Ordinarily a staging is built on two pieces of lagging spiked to the four posts of the two sets nearest the face so as to have the direction that studdles would have. Nails of 30- or 40-penny size are used and are left in either the lagging or the posts when the staging is torn down. They are a constant nuisance and source of danger, tearing the clothes and skin of the miners.

Captain James Rosewall, of the Harold mine, an Oliver property, devised a method of eliminating this trouble. He uses four hooks or hangers of about the design illustrated in Fig. 178, driving them into the post and in them laying two rails or poles on which the staging is built. Since there is danger that the set next the face will fall out toward the other and there is no studdle to prevent, 6-ft. staples, also illustrated, are used to drive into each pair of posts in the direction of the studdle. Since a set of this nature will last as long as a mine, it follows that its

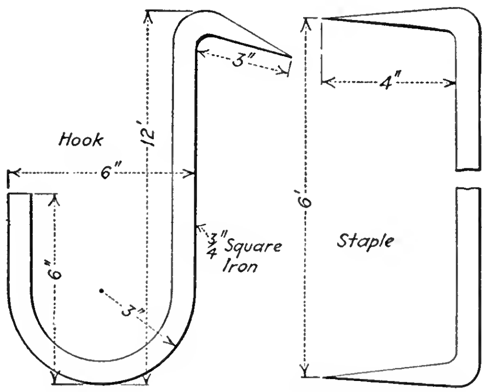


FIG. 178.—HOOK TO SUPPORT STAGING AND STAPLE TO STAY SETS.

use is cheaper in the long run than that of nails. The staple is long enough to span the greatest distance between adjacent sets and by setting it at an inclination is available for intervals down to 4 ft. It is found advisable to bend the drive points of the hooks down from a right angle, as shown, since thus there is no tendency for the weight of the staging and men to pull the hook out of the timber. The pieces are shown made of ¾-in. square iron; round iron may also be used; sharp points are rather preferable to chisel edges on the ends.

Building High Stages with Ladders.—Where the walls of underground workings are strong enough so that they can be left open for long periods of time, the back is liable to become bad and slab off; this is true even where careful trimming has been done before the ore was worked out. It then becomes necessary to erect stages and bar or blast down the bad ground. Usually where there is not much trimming, the back can be

cleaned most satisfactorily from stages built up with ladders as legs. This is the method used in Michigan at the hard-ore iron mines and at those copper mines where the lodes dip steeply.

In order to get the most stable stage, one or two ladders should lean against one wall; but a satisfactory substitute consists of standing three, four or even six ladders vertically on an even base of ore and building a pile of boulders 2 or 3 ft. high around the bottoms of the ladders. By wedging the main stage planks between the legs of the ladders the whole is tightly tied together.

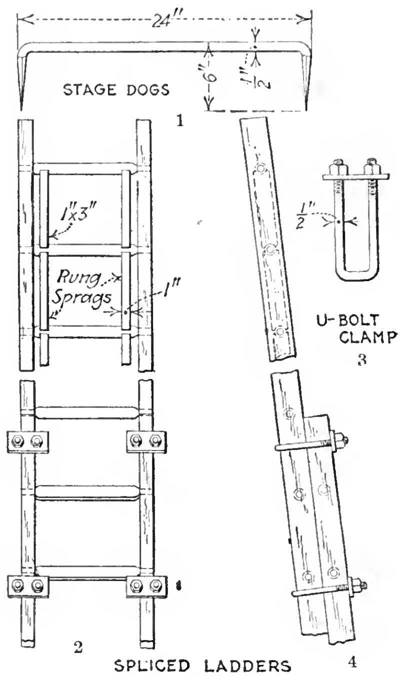


FIG. 179.—DEVICES APPLIED TO LADDERS USED FOR STAGING.

Instead of wedges for binding the ladders to the main or strap planks that carry the planks of the working platform, staples or timber dogs are used at the mines of the Cleveland-Cliffs Iron Co. These are made of ½-in. iron of the form and dimensions shown in 1, Fig. 179. The advantage of these over nails is that they tie the planks together more securely and also can be taken out more easily by means of a pick or bar. In order to distribute the strain over several rungs, pieces of 1 × 3-in. plank with a notch at each end, cut just long enough to fit snug, are driven in as sprags next to each leg between several sets of rungs, as shown in 2. In this way danger from the rungs of the ladders giving away under

the weight of the stage is minimized. Often a stage has to be built to reach a back 30 ft. high or more. As mine ladders are rarely over 16 ft. long, it then becomes necessary to splice them together. Side cleats can be nailed to the legs for this purpose, but a much more secure and handy method is by the use of U-bolts, shown detached in 3 and applied in 4. The bolt is made of ½-in. iron with a plate or double washer of ½ × 1½-in. iron.

Hook for Hauling Timbers.—The accompanying illustration, Fig. 180, shows a simple hook that may be quickly attached or detached from the hoisting rope in moving timbers through a drift or up a winze. The device consists of hook and link, the back end of the hook being enlarged and toothed, so as to act like a cam with reference to the hole for the link. When the rope is passed through the link and the hook is straightened by the load, the toothed cam seizes the rope and pressing it against the link holds it tight until the load is released or the hoisting rope is slacked off. This hook can be attached or detached quickly and in some places will prove to be more convenient than the slower method of tying hitches or knots.

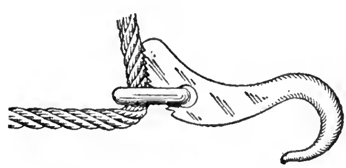


FIG. 180.—ROPE-GRIPPING HOOK FOR HANDLING TIMBER.

Concrete Bulkheads for Pillar Extraction (By Temple Chapman).—The soft-ground mines of the Joplin district are worked by a system which involves leaving a large portion of the ore in the form of pillars to be extracted later. The profitable extraction of the pillars is a serious problem. From one soft-ground mine as an example, where the ore remaining in pillars was believed to be equal in value to that already taken out, the profit on the half of the orebody first mined was $200,000, while on the second half of the ore, that which was left in pillars, it was only about $20,000. The reason for the poor profit from the pillars was due to several circumstances: The caving of the ground mixed together pay ore and waste rock; heavy and costly timbering was made necessary in mining the pillars because of the weight and pressure of the caved ground; much rich ore was lost in the caved mass of ore, soapstone, rock and mud. Big pens of oak logs, built at heavy cost, were crushed flat and mashed like matches by the enormous weight of the ground. To have sunk the shafts deeper and drifted out under all the pillars through barren rock would have cost thousands of dollars and would have re-

covered only part of the ore, and that mixed with much waste, mud and rock.

In the Longacre-Chapman mine at Neck City and the Century mine adjoining, a situation existed similar to that just described, about half of the ore being left in big pillars of considerable value. To attempt extracting these with no other support than ordinary timbering and penning would have allowed the ground to cave, jeopardizing both the miners and the ore. Log pens and timbers where the drifts are 30 to 40 ft. high, have generally failed to hold up the ground after the pillars have been cut out. Nor are log pens of such size at all cheap to put in. The cost of both logs and labor counts up fast. Therefore, an attempt was made to support the roof in the following manner: Holes were drilled between pillars from the surface of the ground to the roof of the drift below, 6-in. drill casing being left in each hole. Forms for concrete, about 15 ft. square, wired across, were being built under each drill hole. Tailings and water happened to be conveniently situated within a few feet of the tops of the drill holes and cement bought in carload lots was distributed at the rate of 200 bbl. for each pen.

A contractor, with a gasoline-engine-driven concrete mixer, mixed the concrete on top and poured it down the drill holes. A man stationed in each form below spread and tamped the concrete as it came down the hole. Empty powder boxes were set in the concrete near the top, several on each side; these served as hitches for timber caps reaching from one concrete pen to another. These timbers caught up the roof between pens and could be additionally supported by posts set under their middle points.

There have been built six of these pillars. They average 40 ft. in height and 16 ft. square. Sixty-pound T-rails set in the cement were carried from the top of one pillar to the top of the next pillar, about 25 ft. away, in addition to the timbers set in the hitches. Cordwood was filled in above the timbers and the T-rails, the latter being braced, thus making a wide support, both the roof over the pillars and the roof between them being held.

Old screen jackets were cut into slices 10 ft. long by 6 in. wide, and used for reinforcement. These strips were laid in the cement east and west, about 6 in. apart. Then a foot higher up they would be laid north and south. The piers increased about 6 ft. in height each day, the previous day's mixture being pretty well set by the next morning. The boards for forms were all recovered after each pillar was completed. When the pillar got up to 35 ft., or within 5 ft. of the roof, the roof was carefully trimmed and left a little high where the drill hole came through. The last cement would then be mixed a little richer and wetter and would fill every space up tight to the top.

FILLING

Sand Filling at Cinderella Consolidated (Institution of Mining and Metallurgy).—The system of sand filling of stopes, devised by Mr. Girdler-Brown, of the Cinderella Consolidated mine, in the Transvaal, compares favorably in first cost with any other method, no dewatering cones or neutralization process being necessary. It shows to greatest advantage when employed in shafts of great depth and under circumstances such that continuous filling is not necessary, as interruptions are almost certain to occur from time to time in wet weather, due to an excess of moisture in the sand.

The sand used should not contain over 5 to 6 per cent. of moisture, and should have been exposed to the sun and air for at least two days before being used. It will then be practically free from cyanide and neutral in character. Sand taken directly from the cyanide tanks was tried for this process, but even after it had been treated with potassium permanganate, considerable quantities of cyanogen were evolved when the sands became mixed with acid mine water. This action was, however, entirely obviated by exposing the sand to the sun and air, as already mentioned. Plans were originally laid out to follow the usual practice in sand filling of running the sand down the shaft mixed with water, but this idea was found to be impracticable, owing chiefly to the excessive wear of the pipe caused by the great depth to which the mixed sand fell and the cost of pumping entailed. When the column first installed was worn out, it was replaced by a square wooden-box launder, down which the sand fell unmixed with water. The launder measured 11 × 12 in. in cross-section, and its cost was approximately 61 cts. per running foot. Observation doors were cut at distances of about 100 ft.

The piping and launder from the surface bins were replaced by a belt which conveyed the sand to the top of the box launder. It was found that sand containing not over 4 per cent. of moisture would run freely from the bins to the belt without handling. On arriving at the head of the launder, the sand falls down the box to a steeply inclined iron plate, over which a stream of water is made to play. The plate should be provided with a liner of the hardest white cast iron to counteract the excessive wear at the point. After being mixed the sand and water flow into a steeply inclined launder, where they undergo further mixture before being conveyed by pipes or launders to the part of the mine requiring filling. The layout is shown in Fig. 181.

The effective capacity of the plant is controlled by the quantity of water available, as it is found that the delivery of the sand to the vertical box is practically without limit. In the Cinderella plant, experience shows that the box launder has not appreciably worn, the reason for

this being the conduct of the sand, which travels normally down the center of the box with little or no impingement on the side. This was proved by examination through the observation doors alluded to. The sand could be seen falling in a steady stream, the bare hand could be held in the corner of the box, but it was difficult to hold an iron bar across the falling sand at the middle of the launder and the metal was quickly polished by the rapidly moving particles. It was noticed that the falling

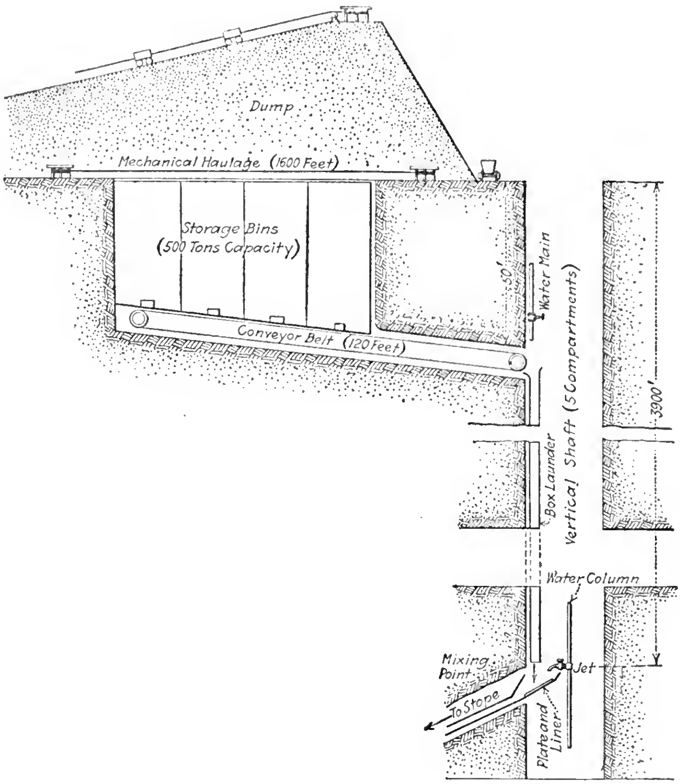


FIG. 181.—LAYOUT FOR DRY-SAND FILLING SYSTEM.

stream of sand created a suction down the launder; thus, on opening an observation door, no sand escaped, but air was drawn in.

From time to time trouble was caused by sand containing too great a percentage of moisture. This caused it to adhere to the sides of the launder in gradually increasing quantities, until at last the flow was seriously impeded. Under such circumstances the remedy was to flush out the box with water from the surface until the adhering sand was washed away. In this connection, experiments were conducted with a view to determining the maximum percentage of moisture which would

allow the sand to run down "dry." The following were the results: Up to 5 per cent. moisture the sand fell freely, leaving the sides of the box clear and dry. From 5 per cent. to 7 per cent. of moisture did not affect the fall, provided that the sides of the box were themselves dry. From 7 per cent. to 9 per cent., the sand gradually became adhering to the sides of the launder where it accumulated slowly. From 9 per cent. up of moisture caused a rapid accumulation of sand along the sides of the launder. The results were largely influenced, it was found, by the proportion of slime contained in the sand.

The liability of the sand to choke the launder under certain adverse conditions renders it essential to have an efficient bell-signaling service between the mining point and the furthest bin, so that the supply of sand can be regulated in proportion to the quantity of water available for service, since if the sand is supplied too quickly, it has the tendency to pile up at the bottom of the box launder and choke it. On the other hand, if the sand appears to be coming down slowly, it may be that a certain proportion is sticking to the sides of the launder on account of there being too great a percentage of moisture. When this is found to be the case, flushing must be resorted to.

Numerous efforts were made to use the current sand production direct from the cyanide tank with a view of saving transportation from the dumps. It was found, however, that this sand, which contains from 12 to 15 per cent. of moisture, gave constant trouble by adhering to the sides of the launder, forming an accumulation. These accumulations happened at various points down the launder, but principally at one point about 600 ft. down. Jets of compressed air were introduced with a view of increasing the velocity of the falling stream, and thus preventing the adhesion of the sand. The box launder was, furthermore, connected with the intake of the ventilating fan near the bottom, and to a Roots blower at the top, the idea being to dry the sides of the box and thus prevent the sand from sticking. These devices undoubtedly permitted the use of damper sand than could otherwise have been employed, but they were practically of no avail when the sand carried over 10 per cent. of moisture, and were consequently abandoned after prolonged trial.

It was found necessary to place the box launder in the upcast side of the shaft and in the same compartment with the pump column; consequently the box was always wet on the outside and water constantly reached the interior. The sand containing not over 4 per cent. moisture does not give rise to any considerable trouble, especially if the launder has its interior surface plain and smooth and the outside tarred. With sand containing up to a maximum of 8 per cent. of moisture, the launder should be placed in the driest compartment available on the downcast side.

There is actually a saving in the quantity of water required to be pumped out of the mine when the sand-filling plan is in operation. The sand in the stope probably retains at least 10 per cent. of water, the sand as sent down contains on an average 3 per cent., and it is calculated that in the course of a good day's run the water saved from being pumped 4000 ft. to the surface will amount to about 8000 gal.

The labor required to operate the plant is small, a subforeman in charge of three boys will look after the belt and surface bins, and the underground part, including the mixing point and the stope to be filled, is in charge of the timberman. The sand is brought from the dump to the surface bin by means of mechanical haulage, the actual shoveling and tipping necessary being done by unskilled labor. On an average of 400 tons per shift, the cost was 5.23 cts. per ton.

Bore-hole System of Sand Filling (*Journ.*, Chemical, Metallurgical and Mining Society of South Africa).—The transfer of sand filling underground through bore holes has proved successful on the Rand in two mines, the Robinson Deep and the Simmer & Jack. The essential features of the system at the Simmer & Jack consist in mixing the sand residue, immediately after car discharge from the vats, with water and a small amount of permanganate of potash solution; and pumping the mixture through pipes to dewatering diaphragm-cone classifiers placed immediately above the bore hole or other point of lowering, down which the thick sandy underflow continuously descends. The fluid cone-overflow gravitates or is pumped back to the mixing box beside the pump, into which the residue is dumped, thus completing its circuit and serving to transport more residue. The underflow falling down the bore hole into the mine is then conveyed by launder to the stope to be filled, where it speedily drains, leaving a solid mass of sand behind. Any accidental filling of the hole with sand is of temporary inconvenience only, as the turning in of a small stream of water at the top in the evening will result in a clearance by the following morning.

Distinctive features of the system are: That it is applied to current residue; that the residue is transported from the sand plant to the lowering point as a flowing pulp and not by cars, the water performing a circuit; that the lowering proceeds continuously instead of intermittently, and is usually performed by passing a thick pulp through a bore hole instead of a more fluid pulp through pipes, thus avoiding wear of the latter; and that distribution of the sand underground is carried out in open launders instead of in pipes under pressure. One portion of the Simmer & Jack mine requiring filling happens to be nearly below the sand plant; so an 8-in. hole 500 ft. deep has been put down near the residue car track, and a short, steep tunnel from the mixing box to the hole allows the residue there dumped to be carried by a small stream of water as a thick pulp

into the mine, without the need of pumping and dewatering. This modification has been more fully developed on the Robinson Deep.

The main features of the surface arrangements at the Simmer & Jack are clearly shown in Fig. 182. This outcrop mine extends over a large area and has been worked for many years. It was hence desirable and economically practicable to lower sand at several points, and the contingency of employing other points in the future had also to be borne in mind. Under these conditions part of the sand from the residue cars is periodically gravitated as a pulp from the mixing box *F* down the tunnel

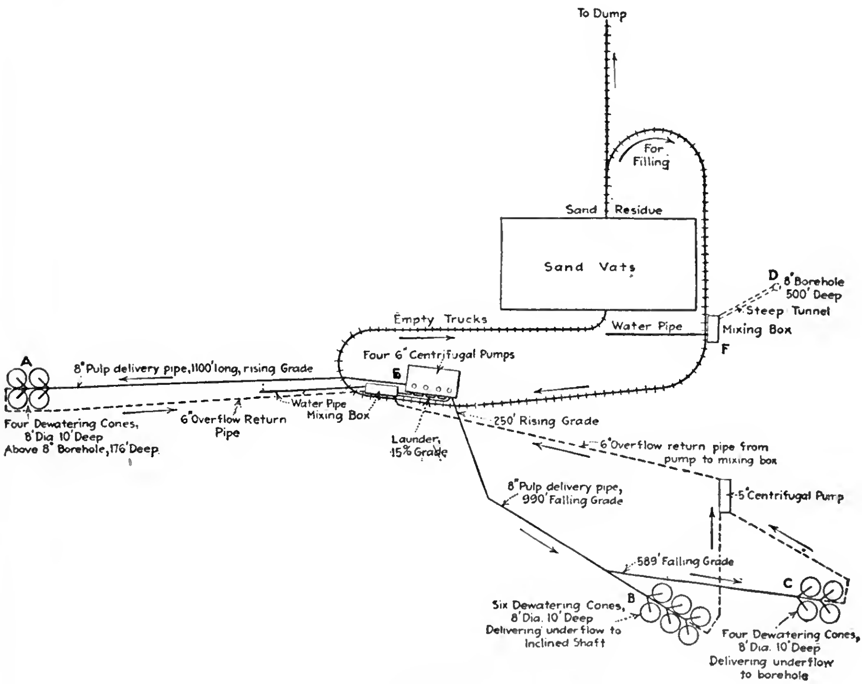


FIG. 182.—SURFACE ARRANGEMENTS AT SIMMER & JACK.

and hole *D*, put down for this purpose. The remaining sand is pumped as a pulp to points *A*, *B* or *C*. From *A* the cone overflow gravitates back to the mixing box *E*, 15 ft. long, 8 ft. wide and 6 ft. deep with a steeply inclined bottom, beside the residue track, although the tops of the cones at *A* are at ground level to saving pumping head. The cones at *B* and *C* are above ground, as their location necessitates return by pumping of the overflow to the mixing box. The graded launder about 20 ft. long between the mixing box at *E* and the pumps, serves as an automatic regulator of the consistency of pulp entering the pumps and prevents the latter from being choked, as extremely thick pulp will not

flow down this launder. The empty cars in every case return to the sand plant.

As the local conditions at the Robinson Deep are different, it was decided after consideration to modify the system somewhat. The greater depth and smaller area of the mine rendered it both advisable and practicable to restrict permanently the lowering to one point at the north of the property, Fig. 183. A bore hole, decreasing from 10 in. diameter at the top to 7 in. diameter at the bottom and 1729 ft. deep, was accordingly put down so that all worked-out stopes could be served from it by pulp gravitation underground, and as the upper end of the bore hole was 18 ft. higher than the track under the sand vats and a considerable distance away, it was decided to drive a 4½ × 6-ft. tunnel from the sand plant at a dip of 20°. This tunnel was 1125 ft. long and intersected the bore hole at 390 ft. from the surface, so that a thick pulp could be gravitated

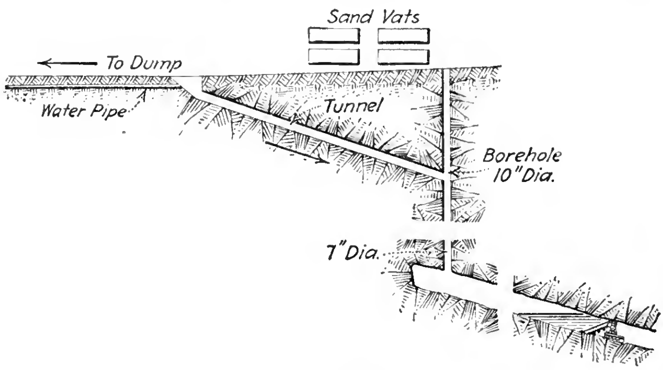


FIG. 183.—SECTION THROUGH TUNNEL AND BOREHOLE AT ROBINSON DEEP.

direct from the sand plant through the tunnel and bore hole into the mine without the need of pumping or dewatering the pulp. The open end of the tunnel is directly under the residue track and just clear of the vats so that the residue after dumping from trucks, and receiving the addition of a small stream of water and permanganate solution, descends as a 28 per cent. moisture pulp till it comes to rest in the desired stope underground. This installation cost a good deal more in capital expenditure upon the tunnel than the Simmer & Jack installation of surface pumps, pipes and cones for a lowering point, but is preferable under the conditions indicated and has the advantage of low surface operating costs, these being merely the short car transport of residue from vats, tipping and returning the trucks, and the addition of a little permanganate. A screen of vertical grizzly bars across the tunnel prevents the bore from being choked by foreign substances. The amount of drainage water to be pumped to the surface from underground involved in sand filling

with so thick a pulp is small, amounting only to about a quarter of a fluid ton per ton of dry sand deposited, or 50,000 gal. daily, with a monthly ore tonnage milled of 60,000 tons.

In the operations of sand filling, the permanency of the underground barriers must be considered. If there is a possibility of water finding its way into the filled area, it will not be sufficient to construct a barrier to last little longer than the period of filling and drainage. Such water, except perhaps in the case of flat workings, would in time wash down the sand through any perished parts of the barrier into the lower parts of the mines. The construction of a permanent barrier is costly, but on many mines dikes, faults and unpayable bodies of ore can be utilized so as to serve as natural barriers. On the Simmer & Jack, every advantage is taken of these natural barriers, and areas requiring filling are often considerably extended so as to permit of their use. It sometimes happens that immediate use cannot be made of such a barrier, owing to the fact that all the ore has not yet been removed from the area ultimately to be filled. In a case of this kind, where there is urgent necessity for filling a portion of the area, a temporary barrier of cheap construction may be erected, enabling filling operations to be carried on immediately.

The number of openings due to previous mining operations through these natural barriers is not usually great, enabling large areas to be completely closed at comparatively little cost. For some time the general practice on the Simmer & Jack has been to close up the drift or other openings with a masonry wall provided with drainage pipes. More recently such drainage has been effected through a bed of sifted clinker resting on a perforated platform. It has been suggested that in order to make this drainage still more efficient and permanent, a system might be adopted consisting in starting a drain with coarse material, such as rock 9 in. in diameter, and gradually working up with rock of diminishing size and finally with clinkered ash until the fineness of the material to be drained has been reached, Fig. 184. This system has everything to commend it in the way of economy of construction, permanency and efficient drainage. The drainage just described is supplementary to the main drainage, which is effected by means of drainage launders. The drainage launders originally constructed were of wood or iron framing, square in section, covered with cocoanut-fiber matting. These were found to be not only expensive but liable to collapse, owing to the perishable nature of the matting. The greater portion of the sand has been drained through ordinary square-section box launders perforated with holes. These launders are cheaply constructed, but have two disadvantages. They require constant attention in order to prevent clogging up by sand before the plugging can be accomplished and they are liable to collapse, although, of course, not to so great an extent as the cocoanut-fiber matting launders.

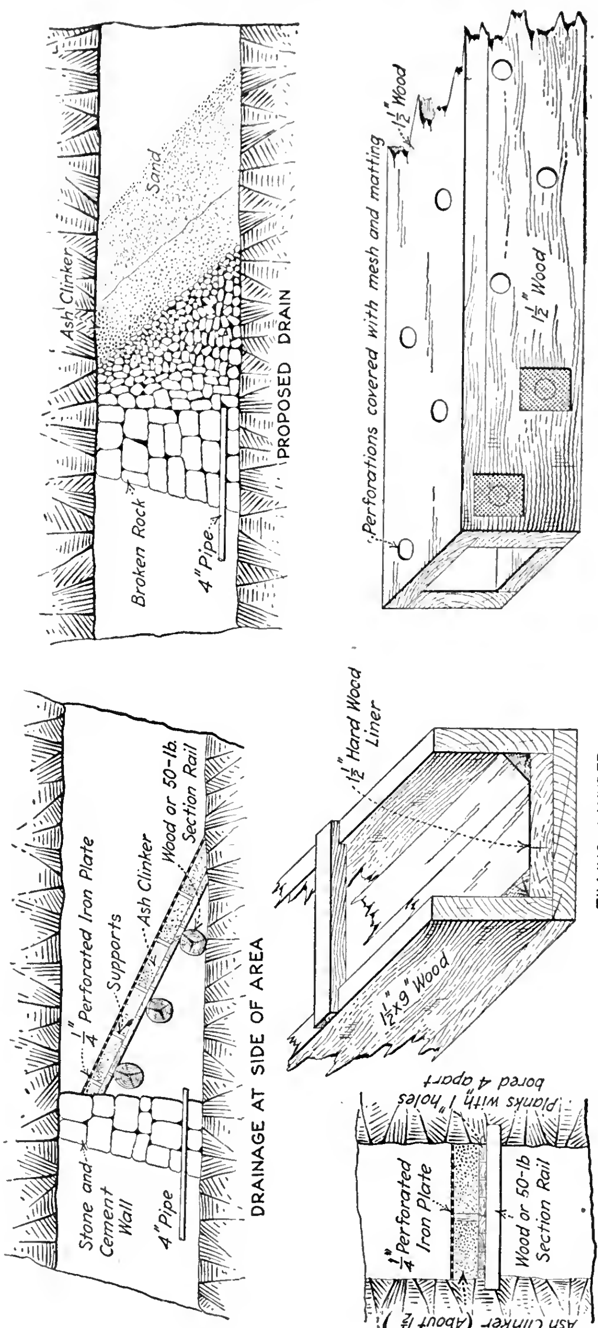


FIG. 184.—DETAILS OF UNDERGROUND LAUNDERS AND BULKHEADS.

It is now proposed to employ a stouter box-shaped launder of smaller internal area. The perforations will be covered with cocoanut matting placed over wire meshing. It has been suggested that these might be further strengthened by filling with a core of sifted clinker. Where the drainage at the bottom of the areas is particularly good and the filling material clean sand free from slime, it is possible to dispense with the drainage launder.

Little change in the form of the filling launder has been made on the Simmer & Jack since sand-filling was first started. For a time wooden pipes were employed, as they recommended themselves both on the score of cheapness and the facility with which they could be erected. The use of these has since been given up and the launder at present used is stouter than originally employed and wears better. It is also deeper in proportion to its width so as to prevent overflow of pulp. The bottom is lined with hard wood and the corners provided with hardwood fillets, which can be renewed when worn. In future the hardwood lining will be replaced by old rubber belting.

Where the dip of the workings is not sufficient to carry the pulp freely along the launders, water is added from some underground source of supply. This is preferable to using water to thin the pulp at the underflow of the cones or mixing box situated at the top of the bore hole, as the water has subsequently to be pumped to the surface.

The cost of filling per ton of sand residue lowered, varies considerably from month to month, owing to fluctuations in the tonnage, and the expenditure incurred underground in providing for current as well as future requirements. On the Simmer & Jack Proprietary mines during a period of nine months, from October, 1912, to June, 1913, inclusive, 172,535 tons of sand was lowered, an average of 19,171 tons per month at a cost for surface operations of 6.736 cts. per ton and for underground, of 13.108 cts., a total of 19.844 cts. These average costs include a considerable amount expended on preparatory work on areas where a large amount of filling remains to be done. It is probable that the monthly tonnage lowered will be increased soon. These two factors should appreciably decrease the cost per ton of sand lowered in the future.

Bulkheads for Hydraulic Filling (*Bull.* 60, U. S. Bureau of Mines).—In the anthracite regions of Pennsylvania, where hydraulic filling for the underground workings is extensively practised, certain types of bulkheads have become standard for confining the filling to its proper resting place. The design of these will vary chiefly with the steepness of the working, that is, the dip of the coal bed. The particular functions of such a bulkhead are to retain the solid material and permit the passage of water. Certain rules for proportioning the various dimensions are as follows:

The haunch distance B, Fig. 185, should be one-half the width of the opening for flat workings or workings dipping up to 10°; two-thirds the width of the opening for chute workings, those dipping between 10° and 25°; and the same for pitch workings, those dipping more than 25°. The width of opening is designated by A in the illustrations. The props should have a diameter in inches equal to their length in feet for

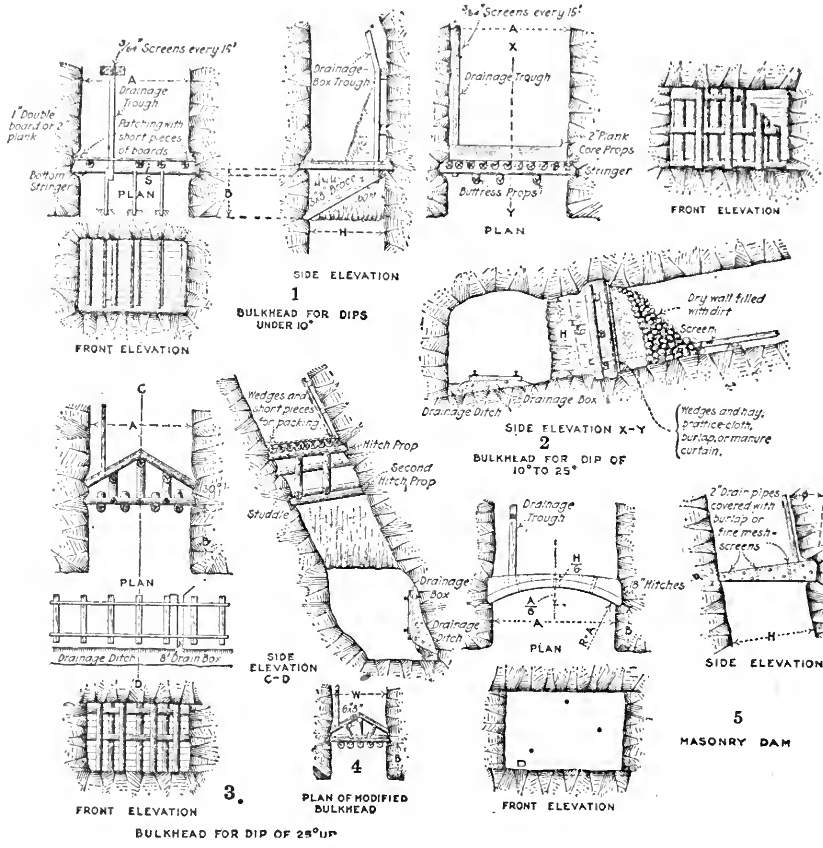


FIG. 185.—TIMBER AND MASONRY DAMS TO CATCH FLUSHED FILLING.

flat workings, one-half greater for chute workings, and three-quarters greater for pitch workings. The spacing of the props across the opening is determined as follows: For flat workings, $S = \frac{A}{H}$; for chute workings, $S = \frac{2}{3}\frac{A}{H}$; for pitch workings, $S = \frac{4}{7}\frac{A}{H}$, it being understood that A = the width of the opening in feet, H = the height of the opening in feet, S = the distance from center to center of props in feet.

For flat workings, the method of construction is shown in 1. The props are securely wedged at the top, and manure or dirt is firmly packed around the bottom after the bottom plank has been attached, to prevent leakage. The inside of the line of props is lined with 1½-in. planks or double 1-in. planks, the bottom, sides and tops being carefully joined or patched by short pieces of board nailed to the main boarding and the solid coal or pillar. For the steeper of the flat workings, a hay, straw, burlap or brattice-cloth packing is used, or a drainage trough is inserted, as shown, the object of either method being to avoid excessive hydraulic head.

The method of constructing a bulkhead for a chute working is shown in 2. In some cases buttress props are advantageous. Drainage should be effected by the use of a trough, as illustrated. The chief advantage of this trough method is the fact that the water is rendered available for use again in the shortest possible period of time. Where the inclination of the work is 18° or more, manure, straw or hay is placed between the two courses of boards as a screener, and in many places a dry wall is constructed to act somewhat as a filter.

A typical bulkhead for a pitch working is shown in 3. The spacing rule in such cases may call for more props than can be inserted in one row, and therefore a second row becomes necessary. The V-form shown in 3 is one intended to resist the highest pressures. The construction shown in 2 is also applicable, using horizontal timbers hitched into the side of the working for reinforcing purposes. Where the V-form is used, a layer of fine manure and dirt is placed as a bedding for the first layer of timbers in the bottom hitches. The planking is placed vertical, with as few nails as possible. In 4 is shown an alternative method of arranging the timbers. The use of a screener and a dry-wall filter is important with pitch workings.

Masonry is frequently desirable instead of timber, the most common form of bulkhead being that of a full-struck arch, with radius equal to the width of the opening, the thickness of the bulkhead at the haunches being equal to one-third of the height, and the crown equal to one-sixth of the height. These conditions are illustrated in 5. Dry walls are also used for bulkheads, and offer the advantage of being both a filter and a retaining wall at the same time. Concrete bulkheads have been tried and found advantageous where great pressures are encountered and timber is expensive.

Conveyor Belts for Distributing Filling (*Proc.*, Australasian Institute of Mining Engineers).—The North mine at Broken Hill was already partly developed on the 800-ft. level when the present management faced the problem of providing an efficient distributing system for filling. Above that level reserves did not warrant the heavy cost of installing

a system to fit in with the mine lay-out then existing. The distribution of filling is here still effected by means of tramming.

The filling consists of residues from the concentrating mill. The main channels for its gravity run consist of three chutes. These are carried down outside the limits of the orebodies, and the filling is conveyed from them on the different levels to the various winzes in the orebodies themselves. A complete distribution of the filling is made to every winze on each successive level, the practice of passing filling from level to level through working stopes being in no place adopted. The advantages of this are many, the chief being that in no place are stoping operations hindered on account of filling operations, and *vice versa*.

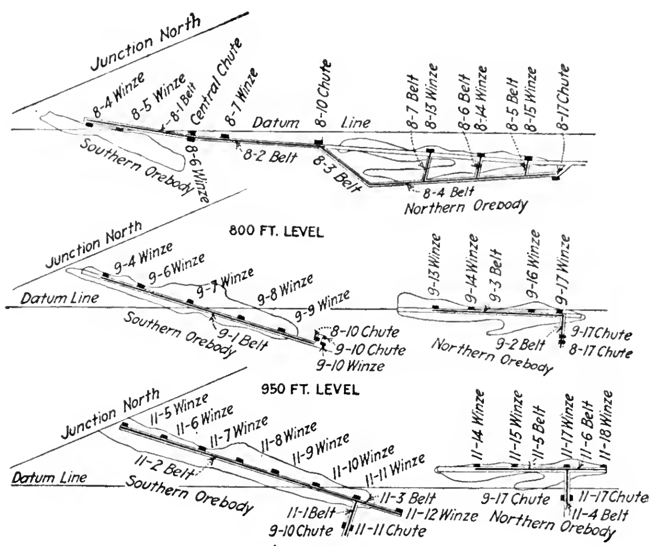


FIG. 186.—PLANS OF BELT DISTRIBUTION ON THE LEVELS.

Of the three chutes, the central extends from the surface to the 800-ft. level, and is continued below that level only in the form of a winze which enters the orebody a short distance below the level; the filling is passed into the stopes off this winze by a direct gravity run from the surface. This chute is situated in the hanging wall of the lode, but down to the 800-ft. level is opposite a barren zone between the two orebodies being worked. The northern limit of the southern of the two orebodies lies just opposite this chute on the 800-ft. level, and to carry the chute down in the hanging might lead to a serious interruption of filling operations should any movement occur in the hanging wall.

A transfer is made on this level to the two other chutes situated 350 ft. and 1000 ft. respectively north of the first chute. Of these the first,

8-10, Figs. 186 and 187, is carried from the hanging wall side of the lode on the 800-ft. level through the barren zone referred to, into foot-wall country at the 1100-ft. level, and below that level will continue in this safe quarter. The second, 8-17, is entirely in foot-wall country below the 800-ft. level. This transfer of the main chutes into foot-wall country places the filling operations in a thoroughly safe condition.

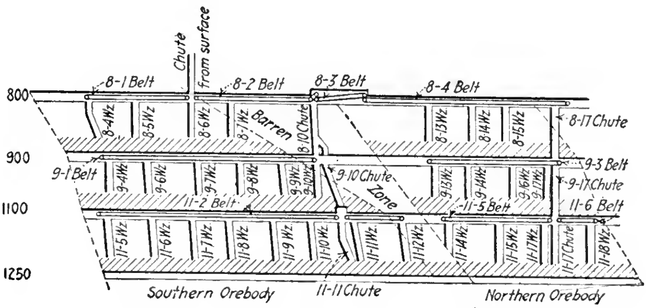


FIG. 187.—LONGITUDINAL SECTION OF DISTRIBUTION SYSTEM.

The transfer of the filling from the central chute to 8-10 chute is effected by means of 8-2 belt, from which filling is also removed into 8-7 winze in passing. The delivery end of this belt is raised so as to deliver on to 8-3 belt, which transfers the filling materials through the barren zone in the lode-channel to 8-4 belt. A temporary chute from which filling was trammed for some time was installed at this transfer station.

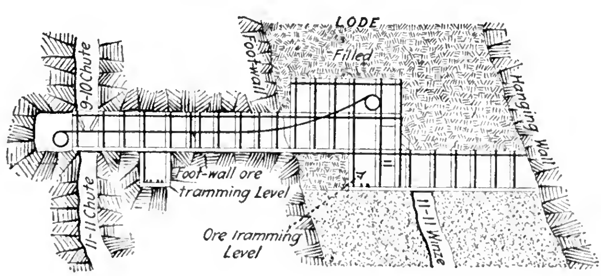


FIG. 188.—CROSS-SECTION THROUGH 11-11 CHUTE, SHOWING SEPARATION OF BELT AND TRAMMING SYSTEMS.

The crosscut containing 8-3 belt is carried over the tramming levels, thus keeping the two operations quite clear of each other. The 8-3 belt delivers to 8-4 belt, which in turn delivers into 8-17 chute. Short subsidiary belts, 8-5, 8-6 and 8-7, take the filling material from 8-4 belt at different points, each delivering into a separate winze in the orebody. From the chute, 8-1 belt delivers into two winzes to the south.

On the 950-ft. and 1100-ft. levels, the plan followed is essentially the

same as shown in the illustrations. A cross-section through the 1100-ft. level, Fig. 188, showing 11-1 belt, illustrates the typical arrangement of transfer from the foot-wall into the orebody, and the manner of keeping clear of ore-tramming operations.

The belts are motor-driven, with a reduction belt drive to a countershaft and a further reduction to suitable speed of driving pulley by gearing. The width of the belt used is 18 in. The belts are run on day shift only, sufficient filling being delivered where required to last through the following two shifts. Colored lights with switches at intervals are provided as a means of signalling to the feed attendant in the event of a change of feed being required. Filling is removed at intermediate winzes by means of rubber scrapers set diagonally across the belt, two being used at each winze; the first removing the bulk of the filling, while the second cleans the belt. When possible, the latter is not used and the belt is allowed to deliver a small amount into the end winze, thus reducing the wear on the belt.

VARIOUS DEVICES

Sconces for Holding Candles.—While acetylene is used in some mines and grease lamps in others, there are many in which candles are used and will be for some time. Mine fires can frequently be traced to a candle source. It is not the candle that the miner is using himself, but the auxiliary lights at some chute, turn or bad place in the run from the ore pile to the chute that cause mine fires. The miner takes home his candlestick but frequently forgets to put out the auxiliary lights. He cannot be expected to provide candlesticks for these, and furthermore, a candlestick is little safer than the two nails driven into a timber, which are usually used. The proper method of holding these auxiliary lights is in a sconce, an old Cornish device. As the name implies, it is simply a candle holder, fastened to the side of an upright timber. To receive the candle, it should have a holder or socket, preferably split so that the miner can easily dig out any old snuff; it should have a back to protect the timber from the flame; and it is also well for the bottom to have a raised lip to catch the drippings. The candle cannot drop out as it burns down, and so fall to the floor. The bottom plate catches the drippings and prevents a large accumulation.

A sconce used in the mines of the Copper Queen company at Bisbee, Ariz., is shown in Fig. 189. This is a cast-iron affair made heavy to resist rough usage. The socket is cast split, and the hole for hanging on the nail is made like a key-hole with the large part down, so that the sconce cannot easily be knocked off. These cast-iron sconces, however, must be made at a foundry; they are also broken frequently at the nail hole. The lighter sheet-metal sconces are preferable as being less liable to

break and capable of being made at any blacksmith shop at odd times out of scrap plate. A good type of sheet-metal sconce is that illustrated in Fig. 190, used at the Highland Boy mine at Bingham, Utah. This is equipped with a handle at the top for carrying purposes. The sheet-iron frame of the sconce bends out at the bottom to catch the drippings, while at the top it is flared to keep the flame from reaching a cap when the sconce is hung high on the post. The candle is held in a socket riveted to the frame, the bottom of the socket being closed so that a snuff cannot fall through.

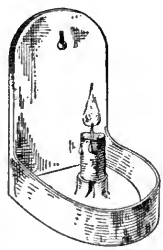


FIG. 189.—COPPER QUEEN CAST SCONCE.

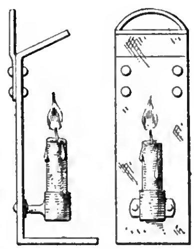


FIG. 190.—HIGHLAND BOY SHEET-METAL SCONCE.

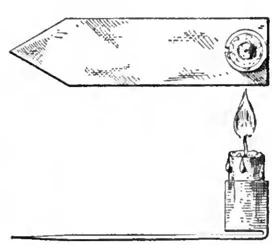


FIG. 191.—CLEVELAND-CLIFFS SHEET-IRON SCONCE.

A simpler sconce is that shown in Fig. 191, devised by Captain Collick of the Cleveland-Cliffs Iron Co.'s Lake mine at Ishpeming, Mich. It is a piece of sheet iron with one end sharpened to stick into the timber and the other turned up and bent into a socket about an inch high to hold the candle. Since it is 8 or 9 in. long, the timber is safe from the flame but the drippings are allowed to fall to the floor. A still cheaper sconce can be made from an old lard pail. The sheet forming the sides of the pail is flattened and the bottom is bent up. The device is held

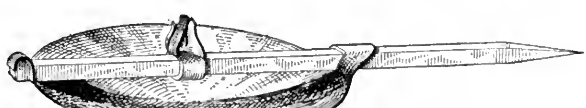


FIG. 192.—HAROLD SCONCE CONSISTING OF SAUCER, STICK AND SOCKET.

to the timber by two heavy nails that also serve to hold the candle. Drippings and the snuff if it should fall out are caught by the trough-shaped flattened piece. In Fig. 192 is shown a candle-sconce used in the Harold mine on the Mesabi. It is of an unusually substantial pattern, both saucer and stick being worked out by the blacksmith. The stick can be driven into a post securely by a rap on the end, this being protected by bending over it a lug of the saucer. The saucer itself catches all drippings.

Ore Chutes of Sheet Steel (*Proc.*, Australasian Institute of Mining Engineers).—Circular steel ore chutes have been used on the South Blocks mine for several years, and continue to give satisfaction under the conditions existing there. The ones first installed were 16 in. in diameter and $\frac{3}{16}$ in. thick, being rolled from 4 × 4-ft. plates. The vertical joint was made by riveting a cover plate on the outside, and the lugs for fastening one length to the one below were also riveted. As these proved too small, some of 20-in. diameter, rolled from 5 × 4-ft. plate were tried, but developed the same fault. Next, some 30-in. diameter by $\frac{1}{4}$-in. were tried, made from 8 × 4-ft. plates, and similar in construction to the others. The size of the latter proved adequate, but the riveting was a source of weakness, as the heads of the rivets got worn off, partly by abrasion, but mainly by the constant jarring of the falling ore. This loss of rivets, in the case of the longitudinal joints, caused the tube to bulge and the chute to hang up. In case the lugs became detached, the tube to which they belonged slipped down and left an annular space, in which the ore again collected and hung the pass up. To overcome this defect the present type of tube was adopted, with no rivets at all in the lugs, and with those in the longitudinal seams placed far enough back from the side of the chute to allow of the jarring effect being deadened.

The sections are made by cutting slots in 4 × 8-ft. plates with a $\frac{1}{2}$-in. slotting punch, the final cut in every case being made with a round instead of a rectangular punch, thus giving a round root to the lugs, to prevent tearing. The circular holes for riveting, and for attaching the slings, are punched at the same time and by the same machine. Provision is made for three lugs on each longitudinal and on each circumferential seam, the bottom longitudinal lug also serving as an additional circumferential support. After cutting, the plates are bent uniformly in the rolls until the opposite edges about touch. The lugs are next bent cold by hand, using a dog, shown in the bottom left corner of Fig. 193, consisting of a 3-in. head, having a slot in it 4 in. deep by $\frac{1}{2}$ in. wide, and provided with a stout iron handle. The lugs of the longitudinal seams are bent out square in one operation, but those for the annular joints are first bent out square to the axis of the tube; then the hold of the dog is shortened, and they are bent back parallel to the axis. The edges of the longitudinal seams are then pulled up together and riveted.

The first section for each chute, or the "starter," as it is called, has the same longitudinal joint, but the lugs are replaced by 4 × 4 × $\frac{1}{2}$-in. angle-iron feet, which rest on the chute timbers and form the foundation for the chute. Riveting is again avoided by attaching the feet in the following manner: Three pairs of circumferential slots are punched

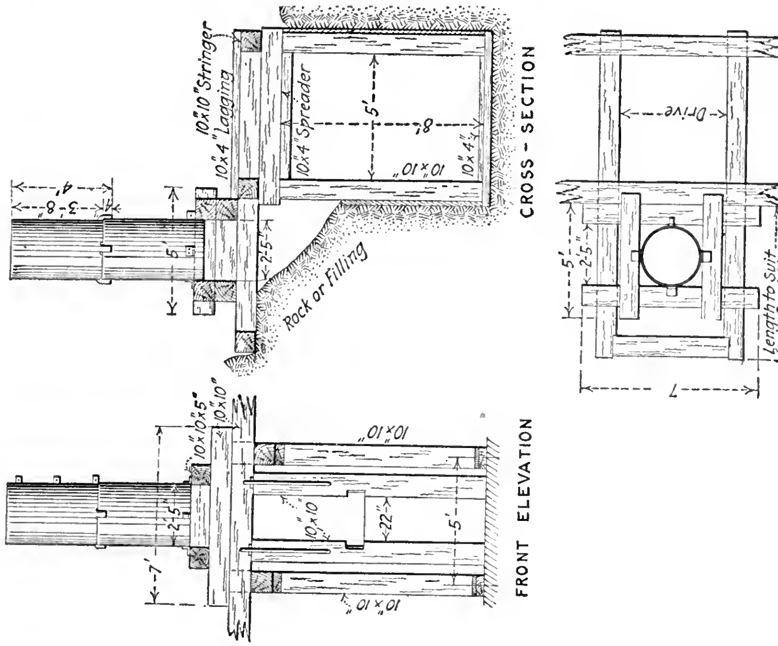


FIG. 194.—METHOD OF ERECTING AND SUPPORTING THE STEEL CHUTES.

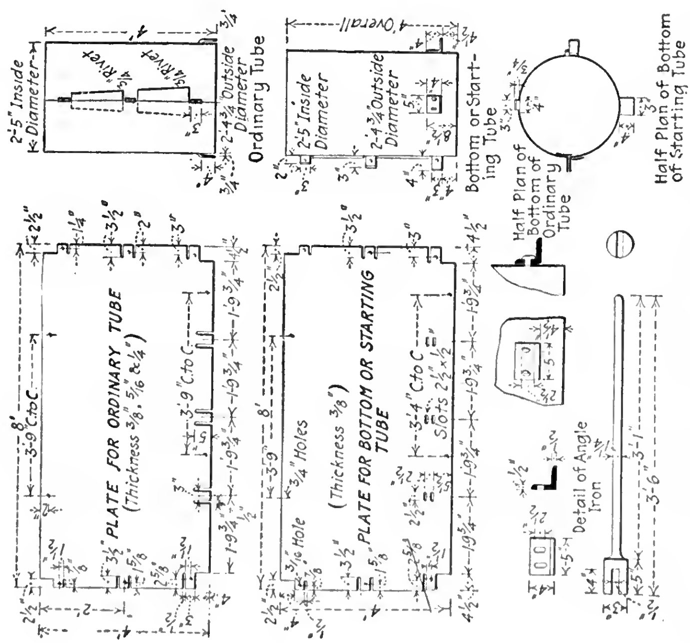


FIG. 193.—DETAILS OF CONSTRUCTION OF CHUTE SECTIONS, SHOWING LUGS AND BENDING DOG.

$2\frac{1}{2} \times \frac{1}{2}$ in., with 2-in. centers; the $1\frac{1}{2}$-in. metal left between is then sheared in the center, and the two pieces of metal forced back to make two $1\frac{1}{2} \times 1\frac{1}{4}$-in. lugs. Over these is placed a 5-in. length of $4 \times 4 \times \frac{1}{2}$-in. angle, slotted to fit the lugs, and these lugs are then hammered back to grip the angle iron. Half and three-quarter sections are also used, being made from 2×8-ft. and 3×8-ft. plates, the object being to suit the height of the filling in the stope. The end dimensions of these short sections are the same as the ordinary ones, but the central longitudinal lug has to be dispensed with from lack of room on the plate. The different lengths of the slots and lugs on the top and bottom circumferential seams, give the required taper to each section. The top of each section has a minimum inside diameter of 2 ft. 5 in., and the bottom a maximum outside diameter of 2 ft. $4\frac{3}{4}$ in., to allow of fitting together. Three thicknesses of plate are used, $\frac{3}{8}$ in. for the first 60 ft. above the level, $\frac{5}{16}$ in. from the 60-ft. to the 110-ft. point, and $\frac{1}{4}$ in. for the remainder; the cost, complete, of each section, from 4×8-ft. plate, is, respectively, \$19.20, \$18 and \$15.60.

In starting from the level, two 10×10-in. Oregon pine foundation pieces of a length to suit the conditions are first put in, Fig. 194, resting on a 2-in. projection of the caps of the level sets, an additional hold sometimes being given by spiking a 4×10-in. piece vertically to the lugs. Two pieces 7 ft. by 10×10 in. are next placed transversely to these and blocked 2 ft. 5 in. apart, and then two 5-ft. pieces parallel to the foundation pieces, similarly blocked 2 ft. 5 in. apart. The starting section is fitted between these, the lugs resting on the top of them, and the whole is made rigid by packing with filling. The chutes are placed 30 ft. apart along the level and in some cases have been taken up 140 ft. above the back of the drift in stopes averaging 30 ft. wide, representing about 15,000 tons handled per chute without any renewals or repairs. Some of the chutes have been lost, but most of these were 16 and 20 in. in size, the rest being ruined by firing with gelignite when clogged. Repairs are difficult, but provided care is taken to avoid sharp bends, which allow the ore to pound out the under side of the bend, and no firing of clogged chutes is permitted, no repairs are necessary where the chutes are properly spaced. The advantages of these chutes are: (1) Their moderate first cost and great wearing qualities which eliminate renewals and repairs; (2) the small size they can be made without an undue tendency to clog, lessening the chance of accident by falling into them; (3) their adaptability to stoping conditions, as no special care is needed when firing ground on top of them; (4) their impervious nature, which prevents any extremely rich ore from being lost, and also confines sand filling to the stope, a matter of great difficulty in wet mines; and (5) the ease and cheapness of installing each section, as each tube is made to fit the one below. Results

on the South Blocks mine show that for the conditions existing there, the steel chute can do its work quite as well as any form of timber chute, and the question of its adoption depends directly on the relative prices of steel and timber.

Chute Conveyor.—A convenient form of shaking chute is shown in Fig. 195. The chains to the end bands are conveniently attached to eye-bolts in the hanging wall. At the end a hole is provided in the bottom for a bolt to connect successive sections. The number of sections

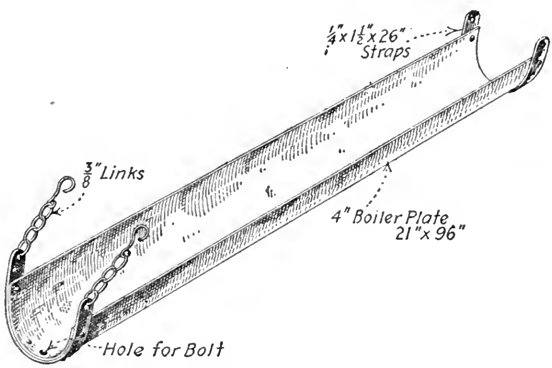


FIG. 195.—SHAKING CHUTE FOR USE IN FLAT STOPE.

is limited by the weight which can be easily swung. The troughs are made of ¼-in. boiler plate, two being obtained from each standard sheet, 42 × 96 in. The straps binding the ends are ¼ × 1½ × 26 in., the chains are of ⅜-in. material. The action of the chutes is obvious. A line of them is hung from the car or bin to the shoveling point near the face and as muck is shoveled into them, they are swung in the direction of their long axis and the material is conveyed by bumping to the lower end. In case it is possible for the muck to slide without bumping, the troughs can be laid on the foot-wall, in a zigzag fashion if desired, so as to reach the shoveling point in the most convenient manner.

VII

TIMBER STRUCTURES

Bins—Chutes and Gates—Skip Pockets—Headframes—Turn Sheaves—Trestles—Ginpoles—Various Devices

BINS

Improved Type of Ore Bin (By Wilbur E. Sanders).—In mining and metallurgical operations, bins are employed for purposes of collection and of distribution, as receptacles within which materials may be collected, wherein reserve materials may be stored and from which such reserves may be drawn as required. The important functions of bins may therefore be summed up as those of storage and the delivery of materials. The two general types of bins are the flat bottom and the inclined bottom.

In Fig. 196, *ABCD* represents the outline of a flat-bottom bin in vertical cross-section. It is essentially self-contained, its frame being supported directly upon the solid bed or sill parts. This type is unquestionably the simplest to construct, the strongest when constructed, and of itself, in all respects save that of immediate delivery at the chute, the most satisfactory, being per ton of capacity the most economically built, reinforced and repaired. When filled, the upper surface of the material will have "coned" to approximately the outline *FGHI*, when the entire cubical contents of the bin will be available for the storage of material. Discharge can take place until a line *EC* is reached, coinciding with the angle of repose of the material, when delivery will cease. The material in the space *EBC*, will remain in place within the bin as reserve or stored material. From the surface *EC* delivered material will be deflected and discharged through the chute. With material thus rolling or sliding upon similar material in its discharge from the bin, the effects of shock and abrasion are minimized. The slope of the inclined plane *EC* will vary with the angle of repose of the material in the bin. Fluent materials will flow readily at a relatively low angle, while others will discharge only at a steeper angle. Material of a clayey wet nature may stick so that only excessive weight of overlying material will force it to discharge and even the bin may become all but clogged.

The material stored as reserve within the triangular space *EBC*, since it can be shoveled out, is available for use in the mill if the supply to the

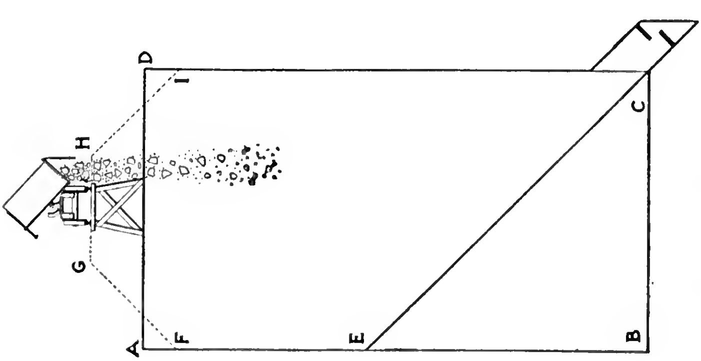


FIG. 196.—OUTLINE OF ORDINARY TYPES OF BINS.

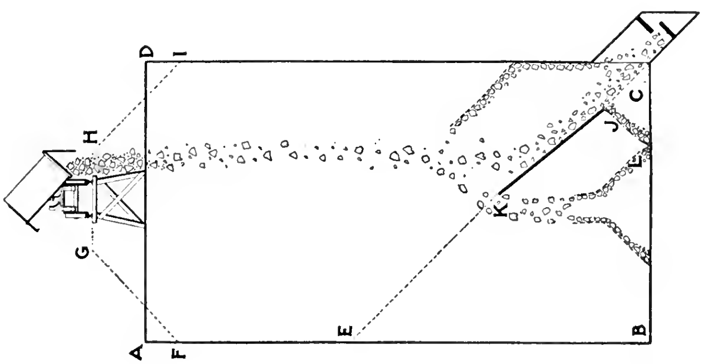


FIG. 197.—OUTLINE OF IMPROVED TYPE.

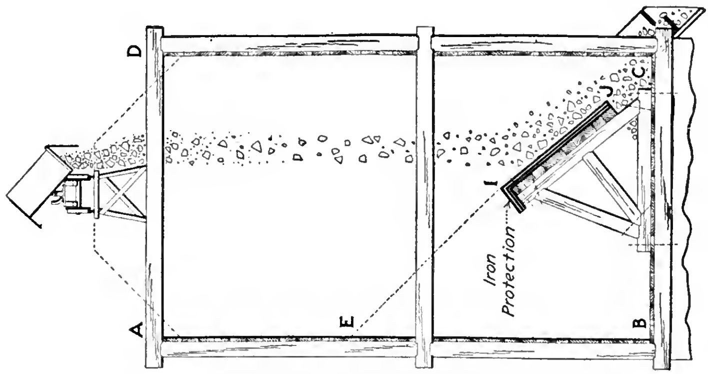


FIG. 198.—CONSTRUCTION OF DEFLECTING LEAF.

bin should cease. This is a characteristic of the flat-bottom bin, of great importance in emergencies. It is counterbalanced, however, by the fact that when such a reserve is once shoveled out, it must be replaced before the b'n can again automatically deliver material.

In Fig. 197, *AECD* may be taken to represent a cross-section of an inclined-bottom bin. The inclined-bottom bin is fundamentally less securely foundationed than is the flat-bottom structure; is essentially unstable, in that it represents a wedge supported on edge. Its construction presents difficulties that can be overcome only by using an ample factor of safety. Furthermore, the excessive weight of superimposed material tends to pack that within the wedged portion of the bin so hard that it presses against the front parts of the structure and even may spring them outward and force them from position. The inclined bottom is also subjected to excessive wear and breakage by falling material and to abrasion by the sliding of the material during its discharge. Repairs to the bottom are consequently extensive and expensive. The large space, represented by *EBC* is wasted, where otherwise it would be available for a reserve of material. The real value of the inclined-bottom bin lies in its facility of discharge, whether it be empty or full. All material that is contained or may be dumped, is available for immediate discharge except for negligible remnants in the corners.

By the application of a simple device, the favorable characteristics of both types of bin may be obtained in a single structure with the practical elimination of their unfavorable characteristics. In Fig. 197 is shown an inclined leaf, shelf or platform, *JK*, within the bin, approximately coincident with or at a slightly steeper inclination than the angle of repose of the material to be handled and placed preferably independent of the bottom, front and rear of the bin. Its lower edge is far enough above the bin bottom to permit material beneath and behind it to be shoveled into the chute. Usually the leaf alone would properly deliver the material, but under some conditions it might become advisable to connect it with the chute by a false or removable leaf; this false leaf may be removed whenever it is necessary to shovel out the stored material. The upper edge of *JK* extends far enough to catch all the falling material from above. If the bin be empty, the falling material will flow back beneath the lower edge of the leaf, forming a pile *JCL*, over the front surface of which any added material will slide to the chute. This means almost immediate delivery of material dumped after the bin has been emptied. If the chute is closed and dumping is continued, the material will cone up on the floor of the bin and on the leaf, and will then overflow the upper edge of the leaf so as to fill the space behind and below it. There is thus no waste space, except that occupied by the inclined leaf and its supports.

The leaf JK can be readily applied to any flat-bottom bin already constructed. Since before any great mass of material rests on the leaf, it will be supported by that which has been forced beneath it, it is evident that it will not have to support any extraordinary weight and need not be of extraordinary strength. It must, however, be so constructed as to withstand the impact of falling material and abrasive action. It may be constructed as an integral part of the bin structure or may be made removable; it may extend the full length of the bin, or may be of a length only sufficient to catch the dumped material and direct it to the chute.

Fig. 198 exhibits one method of applying the leaf to a timber bin; simple angle-frames attached to the sill-pieces by lag-screws support the device. To these frames, spaced at proper intervals, the flooring of the leaf is attached, and this flooring is sheathed either with sheet steel, or

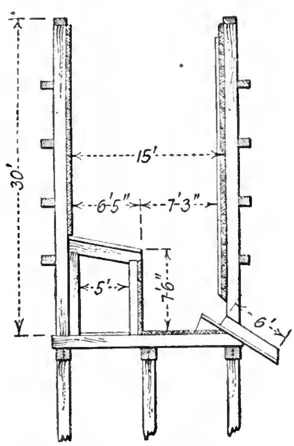


FIG. 199.—CROSS-SECTION OF BIN WITH NEW TYPE OF BOTTOM.

with railroad rails. The surfacing is bolted to the timber floor of the leaf, secured by nuts and locknuts applied from beneath. Thus the leaf may be repaired with little difficulty, since all parts are accessible. Furthermore the wearing parts are of relatively small size and therefore the expense of making repairs is reduced to a minimum.

Bin with Compromise Bottom.—A campaign of reconstruction put through in 1914 by the Ellamar Mining Co., of Alaska, involved the building of a new bin at the mine, with a capacity of 1200 tons of copper ore. Among the main features of this bin are its division into three compartments each holding 400 tons, the placing of all tie rods in the partitions so as to protect them from falling ore and particularly the means adopted to overcome the disadvantages of both the inclined and the flat bottom. For this last purpose a housing was built in the lower rear corner of the

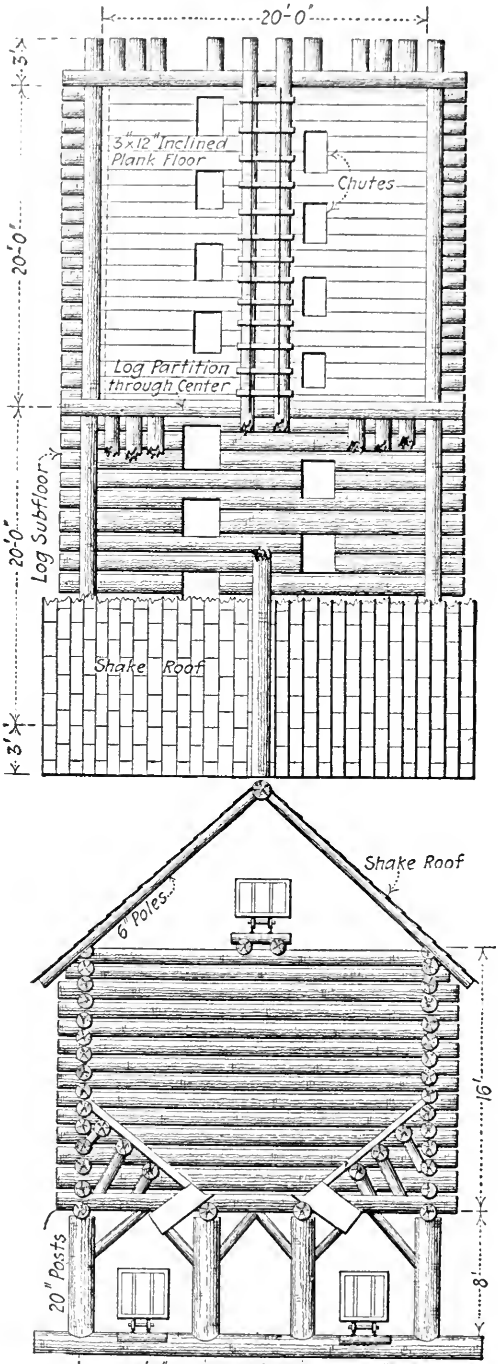


FIG. 200.—PLAN AND SECTION OF ROUND-TIMBER BIN.

structure, as shown in the cross-section, Fig. 199. The space filled by this reduces by so much the quantity of ore tied up and unavailable, a point of importance when the ore is of high grade, while it provides space for storage when desired; at the same time the difficult and expensive construction of the sloping bottom bin is avoided.

A 700-ton Ore Bin of Logs (By W. L. Kidston).—The ore bin illustrated in Fig. 200 was built by W. A. Dickey, at the Threeman mine on Landlock Bay, Alaska, about 5 years ago, and besides being cheap for its size, has shown its stability by withstanding a heavy earthquake shock and showing no sign of strain, although at the time of the shock

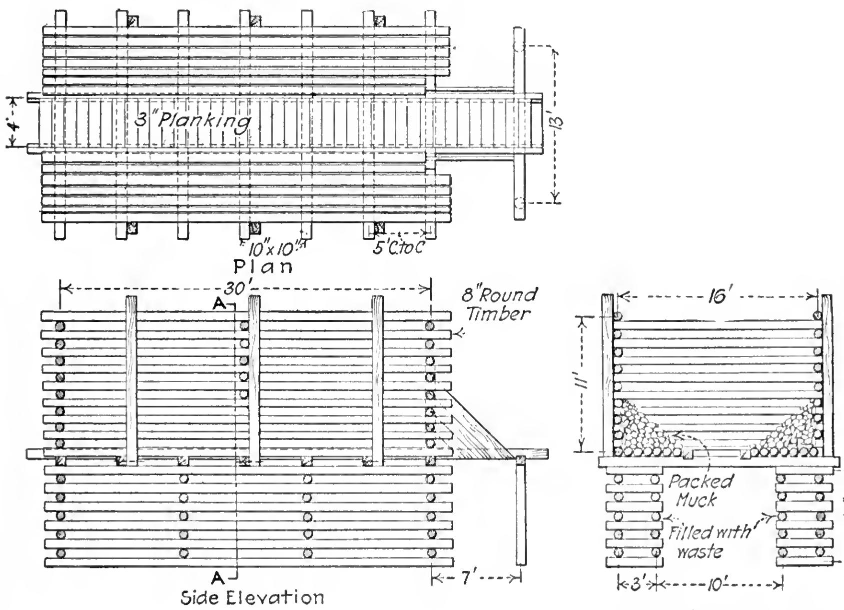


FIG. 201.—PLAN AND ELEVATIONS OF CENTER-DISCHARGE BIN.

one side was full of ore and the other about empty. The bin is built of 12- to 16-in. logs, 24 ft. and 46 ft. long. The large logs cost \$1.50 each, delivered, and the smaller ones in proportion. The logs are notched down just as for a log house and are chinked with cord-wood sticks at \$3 per cord; labor cost \$3.50 per day and the total cost of the bin was \$800. Its capacity is 700 tons. The bin is divided in the center by a course of logs similar to the ends and notched to prevent spreading from interior pressure. The sloped sides near the bottom are planked with 3 × 12-in. lumber. The roof is of shakes. Finger chutes of the Treadwell pattern, not shown in the sketches, are used on the eight discharge openings; the latter are staggered on opposite sides of the center so that

no two are exactly opposite. The ore comes from the mine, a distance of 300 ft., and is dumped into the top of the bin from the mine cars. From the bottom the ore is carried to the ship at the dock 600 ft. distant. Seven men load 60 tons per hour. The pitch of the roof is steep in order that the snow may slide off.

Round-timber Bin with Novel Emptying System (By E. S. Shaw).—A bin on the dumps of Stratton's Independence, Ltd., in the Cripple Creek district has for foundations two long cribs built 10 ft. apart on the surface of the loose rock and filled with waste rock, Fig. 201. Waste from the mine is dumped around the sides of the bin to insure greater stability. The bin timber is chiefly 8-in. round, with the crevices around the sides filled with 3-in. and 4-in. round pieces. The stringers are of 10-in. square material and the flooring of 3-in. planks.

The method of emptying the bin is its novel feature. The car to be filled is run under the first planks that are covered with ore. The loader stands on the platform outside of the bin so as to be safe from falling rock and loosens the first plank, pulling it ahead with his pick. By doing this with each plank in turn until he reaches the ore, he has a space sufficiently large to allow it gradually to fill his car. The ore will run until the sloping face reaches an angle of about 45°, the angle of repose of loose rock. The loosening of more planks provides a greater quantity of material. The capacity of the bin is 220 tons. The entire costs of construction were, for labor, $245; for material, $269.

CHUTES AND GATES

Development of Chute and Gate (By Albert E. Hall).—The chutes here illustrated were tried out at Creighton Mine, Ont., where the ore is a mixture of pyrrhotite and chalcopyrite with a little pentlandite, and is consequently heavy material to handle. Besides being heavy, the ore comes to the chutes in large pieces because the ground has many slips in certain sections and because of the presence of a large amount of rock from a caved section of the mine. These large pieces often have to be blasted in the chutes, so that the latter have to stand hard usage both from the hammering of the ore and from blasting.

Mining at Creighton is carried on by the shrinkage method. The ordinary system of developing a level has been changed, however. On the older levels drywalls were used, while on the new levels rock drifts replace the drywalls. In the drywall method when the point is reached where the chute is desired, the wall is stopped and the chute built and then the masons start the wall on the other side of the chute. In the rock-drift method a raise is put up to a height of from 8 to 10 ft. and the chute built under this. These raises are known as box-holes, and the stope is started from the top of them.

The first chute considered in this method was the Lawson, which had an underswung arc gate, with the concave side of the arc toward the muck. Its general design is shown in Fig. 202. The construction of the chute itself was similar to the Henrotin chute shown in Fig. 207, the only difference being that the pieces marked *A* were made longer in the Lawson chute to support the lip. The gate was operated by an air-lift suspended from above and controlled by means of a three-way valve.

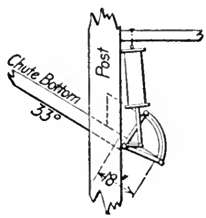


FIG. 202.—LAWSON CHUTE.

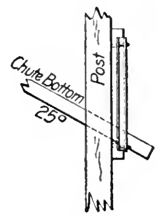


FIG. 203.—FIRST KEATING GATE.

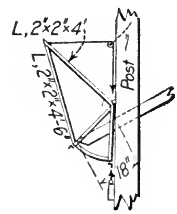


FIG. 204.—COUNTERBALANCED UNDERSWUNG GATE.

It frequently happened that small pieces would get between the gate and the lip or between the gate arms and the lip, thus making it either hard to open the gate or harder to close it, the latter causing a spill in the drift which took a long time to clean up. A remedy was therefore sought, and the result was the Henrotin chute. The gate of the Henrotin chute instead of having the concave side of the arc toward the muck had the convex side toward it but was also underswung. No lip was needed

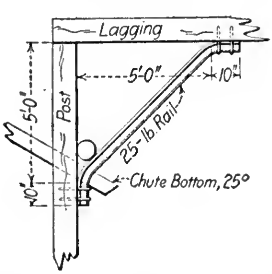


FIG. 205.—MINNESOTA GATE.

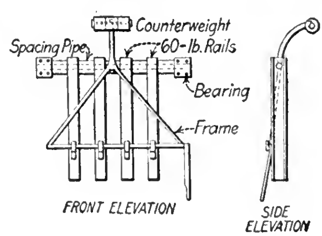


FIG. 206.—FINGER CHUTE.

on the chute because the gate was supported in the side plates, and the front of the gate became the bottom of the chute when it was opened. The air lift was placed below the chute. Putting the convex side of the gate toward the ore relieved the pressure against it and thus made it easier to operate and control. The arc of the gate must be a part of a circle or it will not operate, and one of the difficulties was to get it made true to radius. Trouble similar to that experienced with the Lawson chute

was encountered with this one; fine material got between the chute and the gate and interfered with its operation; fine material also went through on the air-lift below and sometimes stopped its operation. Both chutes had an opening too small for the size of muck that was being handled from the stope, and too much blasting was necessary near the gates, which did not stand up under the heavy duty. This made the cost of repairs high. The air-lifts consumed a large quantity of air, making the operating cost high and reducing the amount of air available for the drills. In winter there was trouble with the air-lifts because of freezing. These facts led to the construction of a chute with a larger opening and without a mechanical gate.

A chute called the Keating, Fig. 208, was then tried. This was a long wide box, which was quickly built and required comparatively

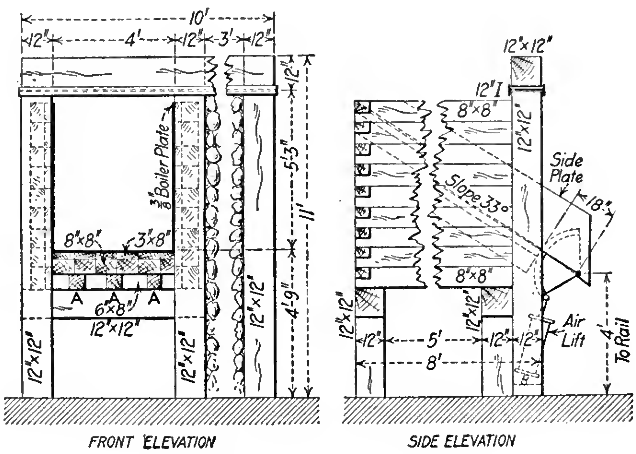


FIG. 207.—THE HENROTIN CHUTE.

little timber. The first of these chutes was constructed with a slope of 15°, which proved too flat; but one of 25° with an iron plate on the bottom was satisfactory. For a gate, a 25-lb. rail was fastened to each post and set out from it, as shown in Fig. 203, so that an iron-bound plank could be run across the front of the chute. This plank was hard to raise on account of the muck behind it and soon became badly bent, when it was impossible to move it. The trammers and chute-men replaced the planks with long drill steel, which worked well, except when fine material ran in the chute; then there would be spills which buried the car and required much time to clear up. When the steel replaced the planks, staples were used in place of the rails. To overcome the difficulty of the fine material running over the steel, a gate, known locally as the Minnesota gate, was tried and has become the regular gate throughout the

mine. It is so satisfactory that it leaves little to be desired. As shown in Fig. 205, two 25-lb. rails with ends 8 to 10 in. long, bent at 45°, are spiked to the chute post and to the lagging over the drift. Across these is placed round timber, which can be raised and dropped easily to control the run of the ore from the chute. The timber becomes worn, but can be replaced with no trouble when the chute hangs up a little.

When the rock-drift method was started the chutes had to be adapted to the box holes. At first a long flat chute was tried, which was a cross between a chute and a shoveling platform. It had low sides and the raise was a long distance in from the chute. The chute-man stood up on the rock at the side and reached over the timber to pull the material

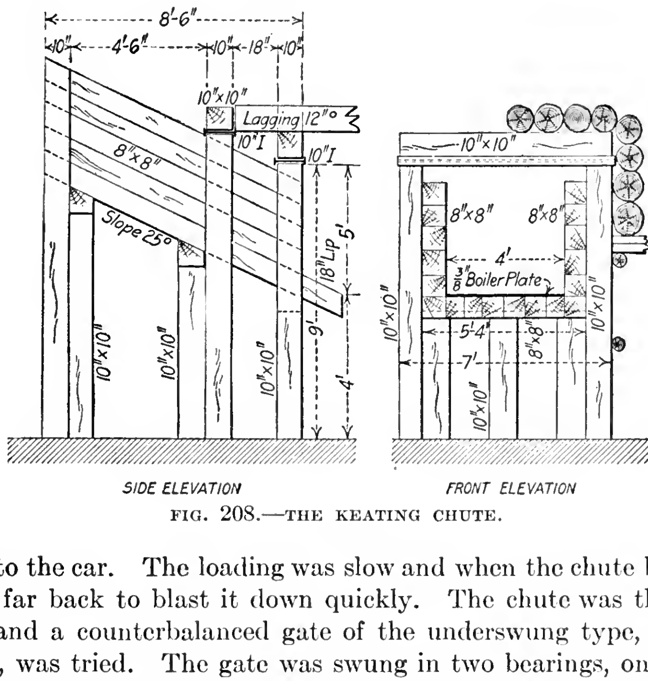


FIG. 208.—THE KEATING CHUTE.

down into the car. The loading was slow and when the chute blocked it was too far back to blast it down quickly. The chute was then made steeper and a counterbalanced gate of the underswung type, shown in Fig. 204, was tried. The gate was swung in two bearings, one on each post, and had its concave side toward the muck. Two angle-iron arms, as shown, were fastened on each side of the gate, one near the bearing and the other near the front of the gate. From the point of each angle-iron arm, a wire ran over a pulley, fastened to each post, to a counterbalance weight for the gate. Although this gate never had a fair tryout, it was, to all appearances, too light to stand the wear, and if made heavier would have been too cumbersome.

An adaptation of the finger chute, Fig. 206, used in other districts was then tried. A shaft running through a bearing on each post supported four 60-lb. rails which formed the fingers. The rails were held

apart by short pieces of pipe. A triangular frame with a handle served to operate the gate, and at the top of the frame were three weights on a short crosspiece which acted as a counterweight. This chute was not successful due to the material handled. It was too heavy for one man to work at times, even with the counterweight, and at other times the material could not be held in check.

The finger chute was therefore discarded and the Keating chute, with the Minnesota gate, was changed, as shown in Fig. 209, to suit the rock-drift system of development. The lower ends of the rails are fastened to the posts and the upper ends are held by staples driven into wooden plugs, which are firmly wedged into holes drilled in the back of the drift. This chute is now the standard for the mine and works well. The back over the chute marked *B* in the figure was at first made from 6 to 8 ft. long, but has been cut down to from 2 to 3 ft., which makes it

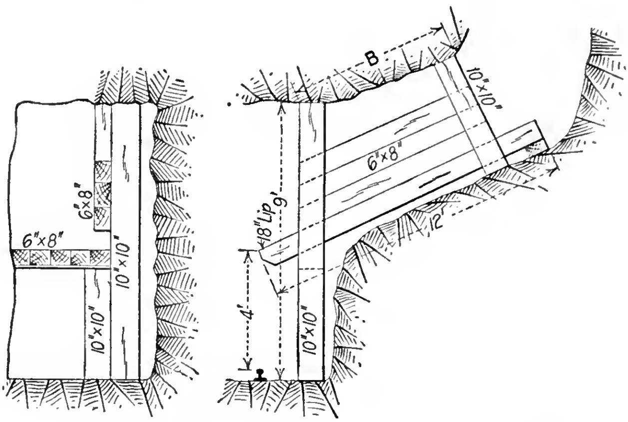


FIG. 209.—KEATING CHUTE ADAPTED TO ROCK DRIFT.

much more convenient for the man barring or blasting. The lip, which is 18 in. long, sometimes catches on large pieces on the top of the cars; but owing to the type of cars used this cannot be remedied. A new type of car is to be introduced which will remove this trouble.

Ore-chute Side Pocket (By Lewis B. Pringle).—In Fig. 210 is shown an ore chute used by the Cananea Consolidated Copper Co., in its Capote mine. The chute was designed by the geological department of the company, and was first used in getting the silica ore from the "silica stope." The chute in itself need not differ from any of the many chutes now used. The feature newly introduced is an elevated pocket on one side of the chute, reached by a short ladder.

Ordinarily, if a chute becomes jammed, the jam must be broken by one of two methods, *i.e.*, either by discharging a piece of dynamite in the chute, or by breaking the jam with a crowbar or "prod." The first

method is frequently objectionable because of the danger of injuring the chute and because of the resulting fumes. When the latter method is preferable, the arrangement herein described will prove to be a time-saver and a help to the man who is using the crowbar. The car-filler generally has either to reach over the partly filled car, or get upon it, and poke his bar up through the chute gate. The arrangement herein described permits the car-filler to climb into the adjoining pocket, reach over the planking shown in the side elevation and get at the seat of the trouble more easily and more quickly than otherwise. Often the rock jams at the elbow of the chute, and the pocket herein described enables a man to reach this elbow, as he could not if working from below.

The erected chutes do not follow the drawing in detail; the latter merely illustrates the idea and the chute builder constructs the chute to meet the required conditions. In building the first chute of this type,

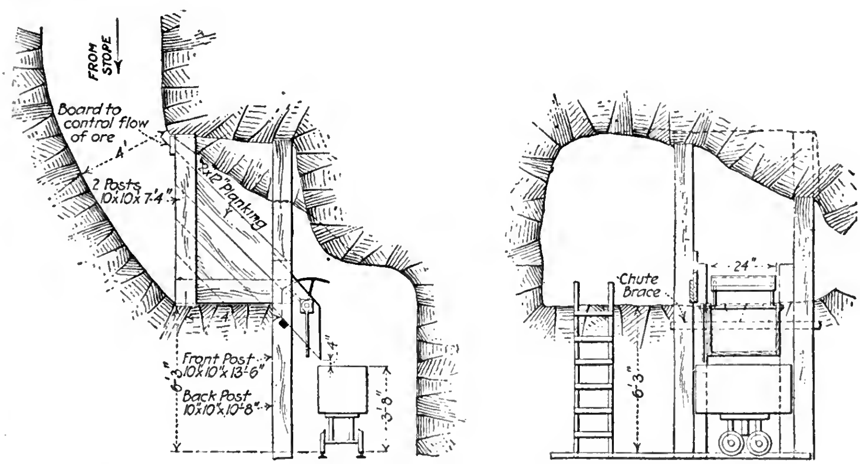


FIG. 210.—ELEVATIONS SHOWING RELATION OF POCKET TO CHUTE.

the sharp elbow shown was cut off, and the back stulls were lengthened a foot or more. The few dimensions given are more or less standard. The planking which keeps the rock from filling into the pocket, can be raised or lowered as necessary, leaving enough room between the top plank and the roof for a man to work over. The board that controls the flow of the ore is also raised or lowered as required.

Bulldozing Chute and Underswung Gate (By G. J. Jackson).—At the Perseverance mine of the Alaska Gastineau Mining Co., at Juneau, mining is conducted in large open stopes somewhat on the shrinkage system. Trouble was experienced from large rocks weighing several tons being drawn into the chutes; these rocks had to be blasted, with the result that chute timbers were often broken, and a continual expense incurred in the upkeep of the chutes. A chute with a special bulldozing

chamber fitted with grizzly bars, was tried early in 1912. It proved a complete success, over 20,000 tons of rock being drawn through it without trouble. Fig. 211 is a section showing the general arrangement. The 80-lb. rails used for grizzly bars are spaced 2 ft. apart, one end resting in hitches cut in the foot-wall and the other end on a 24 × 24-in. cap. Any rock passing through the grizzly can easily be handled in the cars; the larger rocks which remain on the bars are drilled with a small plugger machine and blasted. A ladderway, not shown, leads up to the bulldozing chamber from the level. When the ore is running freely, containing few large rocks, the grizzly bars, over the back pocket, will be partly covered. Should the front grizzly block up, the small rock is got

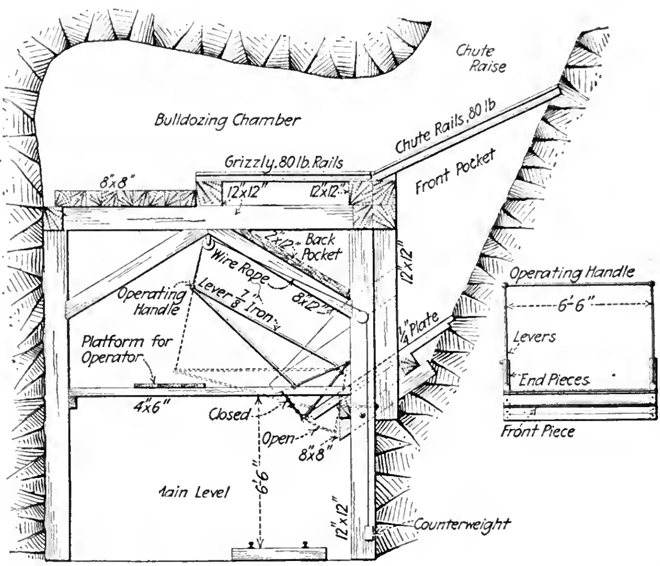


FIG. 211.—POCKET, CHUTE AND GATE.

FIG. 212.—FRONT ELEVATION OF GATE ALONE.

rid of into the back pocket, and the large rocks thus made accessible are blasted. The two pockets together hold about 25 tons.

The arc chute gate shown in front elevation in Fig. 212, is of the drop-down type. It is counterbalanced and is operated easily by one man by depressing the handle. It will handle any kind of rock and will shut off even with a 1-ton rock passing through. The operator stands on the platform above the cars, and is completely out of danger from rocks falling from the chute. A slide gate, not shown in the sketch, is used above the arc gate to check the rock. The arc gate is made of 1/4-in. iron plate, 18 in. wide and 6 ft. 6 in. long, bent with a 3-ft. radius. This is stiffened for its full length by three 2-in. angle irons riveted on. The two triangular

end-pieces are made of ½ × 3-in. iron, reinforced by a piece of the same material fitted inside the arc. The two side levers are of ⅞-in. round iron, connected with a ⅞-in. iron handle parallel with the gate. Two $\frac{3}{16}$-in. wire ropes attached to the end of each lever and passing over small sheave wheels on the chute timbers, support weights sufficient to balance the gate. The gate is made in the mine shop and can be quickly set up on any chute, provided there is sufficient headroom to accommodate the operator.

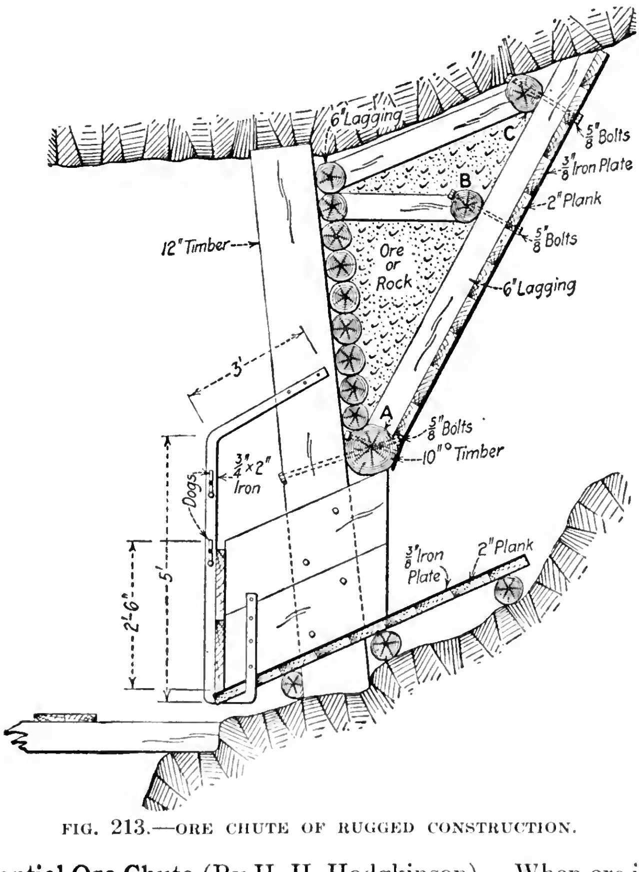


FIG. 213.—ORE CHUTE OF RUGGED CONSTRUCTION.

Substantial Ore Chute (By H. H. Hodgkinson).—When ore is dumped into most ore chutes it strikes a direct blow almost at right angles to the front of the chute. The chute shown in Fig. 213 is so constructed that it receives a glancing blow, rather than the full impact, and will resist a great deal of pounding and banging because it is backed with rock or ore.

The stulls and horizontal timbers *A*, *B* and *C* are placed in hitches and carry the chute and the weight of the ore in the chute. The timber

A is bolted to the stulls as shown, while a layer of 6-in. round lagging is spiked in a horizontal position to the stulls. A layer of 6-in. round lagging is spiked to the horizontal timbers B and C, while the lower ends are supported by the timber A. The lagging is trimmed so that a layer of 2-in. plank which is spiked to it fits smoothly; this insures a good surface upon which to place the 3/8-in. wrought-iron plates so that they will not buckle and tear loose. The iron plates are 4 × 8 ft. and have countersunk holes for 5/8-in. bolts which pass through the main timbers A, B and C, as shown, clamping the back of the chute firmly together, in addition to securing the plates. Iron plates are bolted on the sides of the planks at the mouth of the chute and also to the bottom. The timbers are stulled to stiffen the chute, and the space between the front and back of the chute is filled with ore or rock which gives solidity to resist shock.

Although the initial cost of this chute is somewhat greater than that of the average chute used, its life is many times longer. On a number of

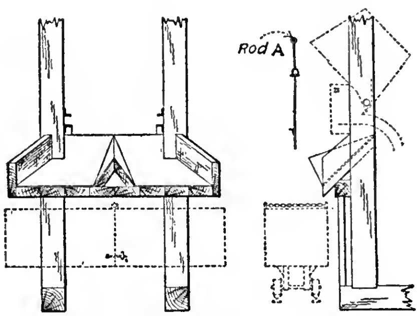


FIG. 214.—DUMP ARRANGEMENT FOR UNDERGROUND HOIST.

them in use constantly for four years and receiving hard treatment no repairs whatever have been necessary and they are as good as when first installed except for a little wear at the lower ends of the iron plates.

Chute for Loading Car from Skip (By W. W. Shelby).—When conditions are such that it is desirable to have a skip dump directly into cars, Fig. 214 shows a satisfactory arrangement for an underground dump. The width of chute is 1 ft. less than the length of car. The middle partition serves two purposes: To distribute the ore so as to utilize as much car capacity as possible, and to protect the chain across the top of the "middle dump" cars. A 3/8-in. iron plate, stiffened by a 1 × 1 × 1/4-in. angle iron is suspended in front of the chute. This plate, the dimensions and position of which depends upon local conditions, swings freely about the rod A, and deflects the few stray pieces of rock which might otherwise go over the car in too rapid dumping.

Hanging Chutes (By L. D. Davenport).—When, in mining by the top-slicing system, the ore has been sliced off down to the last sublevel above the main level, usually about 10 to 12 ft., it is evident that an ordinary chute will hold only about one car of ore at a time. It often happens in such cases that the miners have to wait until the trammers empty their chute before they can go on with their work. To prevent such waits a hanging chute was devised which has been successfully used in one of the mines of the Chisholm district on the Mesabi range. Fig. 215 shows the general construction of the chute. The track from the sublevel is carried by short drift sets, which also support the sides of the chute. The chute bottom is made of loose boards resting on 30-lb. rails which run lengthwise of the chute and carry the greater part of the weight. These rails are hung to the caps of the small sets so as just

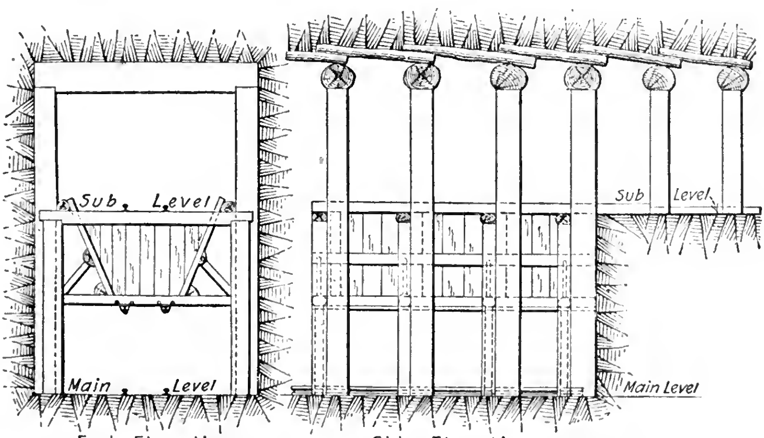


FIG. 215.—HANGING CHUTES USED IN SUB-LEVEL MINING.

to clear the top of the tram cars on the main level. These chutes are sometimes made 20 ft. or more in length and hold a large number of cars of ore. One tram car may be filled, or several at the same time, by loosening and removing a few of the bottom boards.

Concrete Storage Chutes in Stopes (*Bull.*, American Institute of Mining Engineers).—It has been found economical at the Copper Queen mine in Arizona, for handling certain kinds of ore, to put in concrete pockets and cylindrical raises to be used as storage chutes, since the upkeep cost is practically nothing, whereas with timber chutes this is high. Fig. 216 shows a concrete pocket built particularly to handle sticky ores from the Dividend slice. It has the shape of a funnel, with the large end downward. About 30 ft. above the sill, in the top of this funnel a 45° offset or baffle was put in, and from this point the raise was

continued to a level 200 ft. above the sill. This raise is circular and lined with concrete, and it is confidently believed that this type of pocket will materially lower the cost of handling the ore.

Chute Reinforced with Angles (By Albert G. Wolf).—In mining by the shrinkage method, large boulders are often covered in the stopes. These boulders eventually work down to the chutes, and to remove

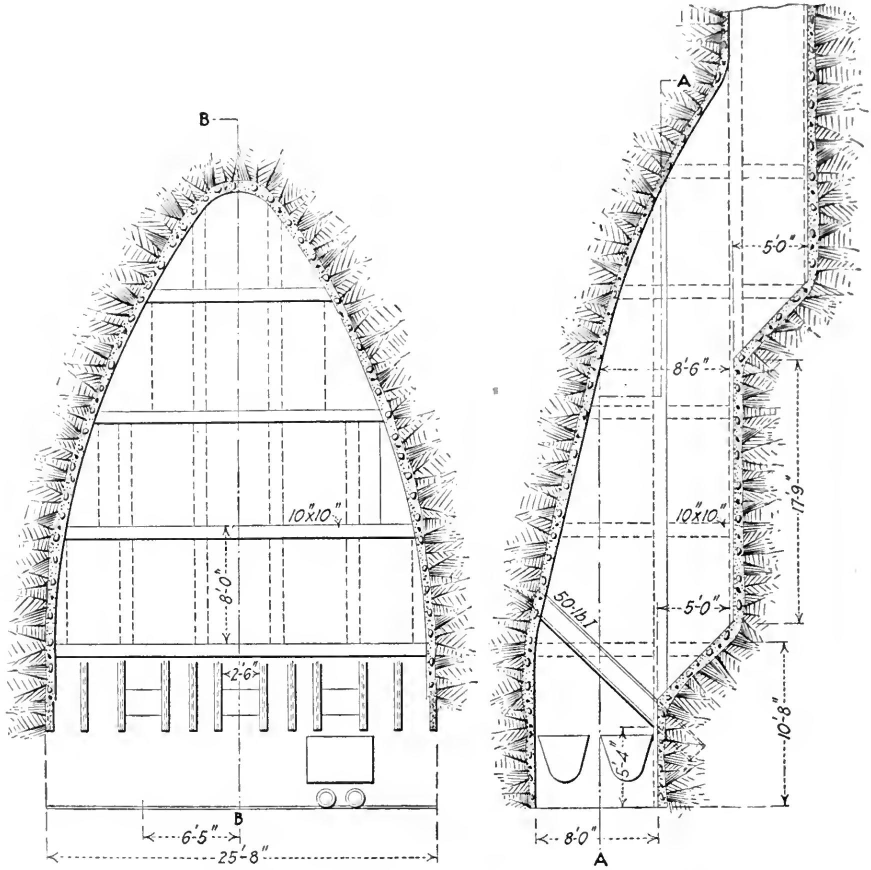


FIG. 216.—VERTICAL SECTIONS OF A CONCRETE-LINED STOPE POCKET.

them it is necessary to resort to blasting, which will weaken or destroy the chute timbers. A simple reinforcement against such injury is shown in Fig. 217. Pieces of $\frac{3}{16}$-in. sheet iron are cut to fit the chute sides, and to this sheet iron are riveted pieces of $\frac{1}{2} \times 2$-in. angles, forming both braces and cleats to hold the chute boards. These sheets or liners are then fastened to the inner sides of the chutes by spiking or bolting. The exact length of the angle braces is immaterial. A piece of

sheet iron covering the bottom of the chute will also add greatly to its life.

Quincy Rockhouse Loading Chutes (By L. Hall Goodwin).—The Quincy method of loading stamp rock from the storage bins at the mine into cars for transportation to the mill is unique among methods used at the Lake Superior mines in that the chute aprons are operated by compressed air. Fig. 218 illustrates the chute mechanism. The air cylinder is direct connected to one corner of an irregularly shaped, four-cornered steel plate, the other three corners of which are connected by levers with the lower apron of each chute and a hand lever, respectively; the plate is pivoted so that the various levers have the right amount of play. The hand lever is, of course, thrown whenever the aprons are

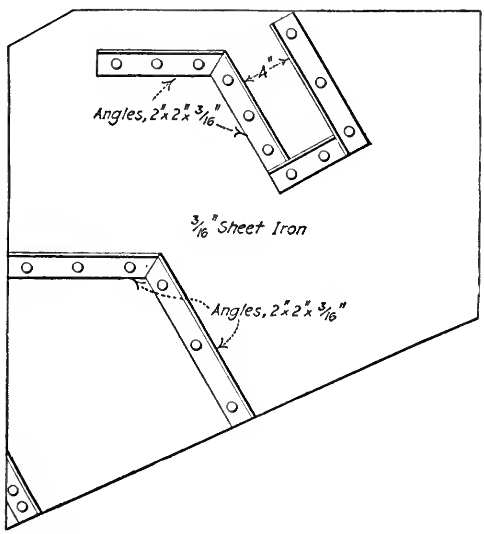


FIG. 217.—STEEL PROTECTION APPLIED TO CHUTE.

operated but it is of use only in emergency, when the air accidentally goes off; the aprons cannot be operated easily by hand, but are opened and closed quickly by the air cylinders. The air cocks are centrally placed in a niche left in the concrete bin foundation midway of the tunnel; this niche is just large enough to allow the operator room to work comfortably. There is one air cock for each cylinder, that is, for each pair of chutes, and they are arranged on the two sides of the operator's niche corresponding to the position of the chutes they control. The air cocks and pipes are arranged to admit air to either end of the cylinders.

The operation of loading is as follows: The string of cars is pushed through the tunnel by the switching locomotive and left there, the grade of the track being such that the cars may be started easily when the brakes are released. Cars are loaded by two men, one of whom oper-

ates the chute aprons; the other rides the forward car and by use of the brake keeps the cars in slow but steady motion, the aim being not to allow them to come to a standstill because they might not start again of their own accord. As soon as an empty car comes under the end chutes the aprons of that pair of chutes are opened and remain open until the car is about to pass from under them; the filling of the car is completed as it passes along under the other chutes, and its load is finally topped off by a few splashes from the last pair of chutes. To accomplish this, several pairs of chutes must often be open at once, and the operator must be alert to avoid spilling rock on the track; his work is not difficult, however, as it consists of simply turning air cocks. Air is supplied from the main underground system; between shifts or at other times

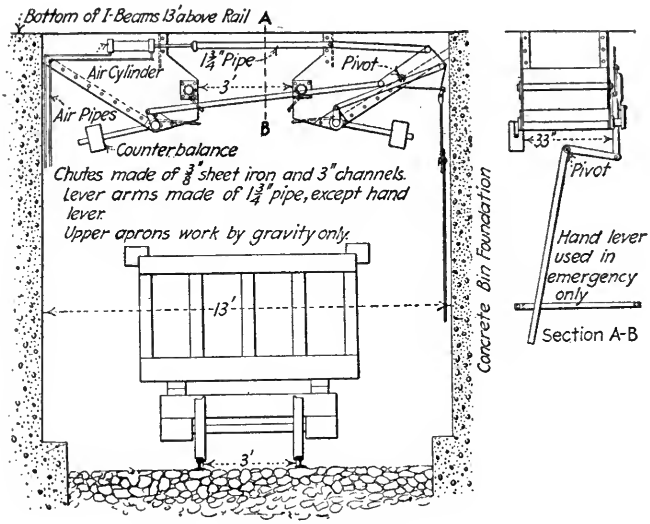


FIG. 218.—LOADING CHUTES AT QUINCY'S STEEL STORAGE BINS.

when the large air compressors are not working, a small Westinghouse air pump takes the load, it being adjusted to start and stop automatically whenever the pressure in the underground system falls below or exceeds a certain amount.

This method permits more rapid loading than does the usual one, with which it is necessary to stop each car for some time under the chutes. This is true not so much because of their operation by air, however, as because the chutes are located in pairs on opposite sides of the track and there are more of them than usual.

Underswung Gate (By J. R. Thoenen).—Fig. 219 shows the front and side views of a chute and gate for use where the bin supplies mixed fine and coarse ore. The chute proper is made of one piece of $\frac{1}{4}$-in. steel

plate, the bottom extending 6 in. into the bin to protect it from wear of running ore. Two wing plates of ¼-in. steel are riveted on the vertical sides to protect the sides of the chute, and for the same purpose, a piece of angle steel may be placed on top of the opening. In this instance, the bottom of chute is 16 in. long, but may be altered to suit the existing arrangement of timbers.

The gate is hung on a 1½-in. round-iron axle supported in holes bored through the sides of the chute. The sketch shows the chute open and the gate down. The top of the gate itself forms the lip of the chute. A timber is placed under the gate, so that when open, the gate rests upon it and is flush with the bottom of the chute. The gate is worked by a short detachable lever (not shown), which is rounded at one end to fit in

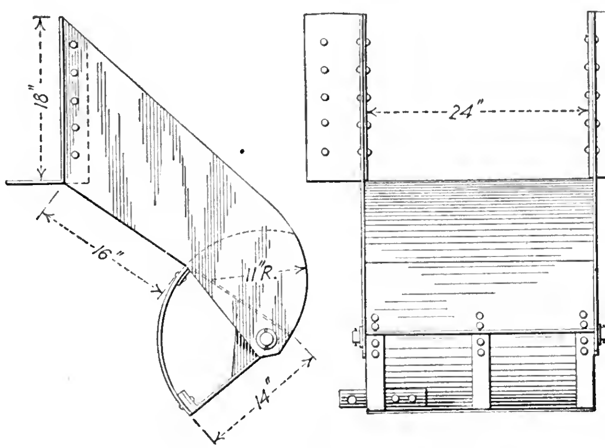


FIG. 219.—STEEL CHUTE AND GATE ASSEMBLED.

the hole in the short piece riveted to the bottom of the chute gate. A hole through the lever fits over the end of the axle and forms the fulcrum. Washers and pins are provided to keep the axle in place. The gate is made of two pieces of ¼-in. steel plate, one bent in the arc of a circle of 1 ft. radius, the other flat with one edge turned over 2 in. at a little less than a right angle. These two plates are riveted together with countersunk rivets. The gate bearing for the axle is provided by a small web piece riveted at each end to the upper plate and lower brace. Three ¼ × 2-in. braces support the under side of the gate. When the gate is well centered, so that the arc forms a true circle with the center of the axle as its center, there is practically no trouble with fines causing the gate to stick.

Supporting Sliding Gate and Lever.—In Fig. 220 is shown the manner of supporting the lever of a chute gate used on some of the chutes in the mines of the St. Louis Smelting & Refining Co., in southeastern Missouri.

The gate lever is carried by two iron brackets bolted to the uprights of the chute and is bent up so as to be out of the way of miners passing along the track; and is then bent downward again so as to be within reach of the miner on the opposite side of the drift. The carrying braces are bent in so that they are separated by a space only as wide as the thickness of the lever bar, at the point where the bolt goes through.

A slide gate is used that runs in an angle-iron guide fastened through the side planks to the main timbers of the chute. The lever is connected by a pair of clevises to the gate, but it also extends far enough beyond the gate to come over it and to shove it down to its seat in case it should

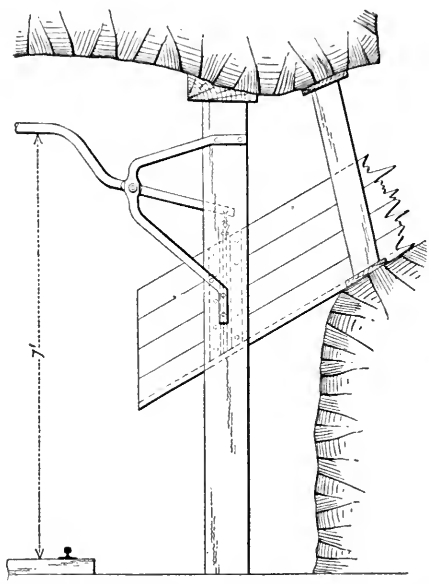


FIG. 220.—GATE LEVER EXTENDING ACROSS DRIFT.

stick. The gate works well and no more "runs" of ore occur than would be the case with any top-closing gate. The manner of supporting the lever allows it to be brought out straight in front of the gate so that the loader can look up into the chute as the ore runs out and shut or open the gate as needed. The bending of the lever up into a goose neck so that over the track it is above a man's head when the gate is only half closed, removes any objection to having the gate lever extend across the drift.

Pneumatic Underswung Arc Gate (By J. R. McFarland).—At the Cactus mine of the South Utah Mines & Smelters, Beaver County, Utah, a pneumatically operated ore-chute gate was used. The gate, as shown in Fig. 221, was operated by moving a lever controlling a four-way

valve. To the bottom corner of the chute was attached the supporting shaft. The door was constructed of a square sheet of boiler plate the width of the chute, bent in an arc with the face up, and supported on the shaft by two legs, so as to revolve about the shaft as an axis. To the side of the door on the underside of the chute, was attached a piston working in a cylinder. The piston rod was hinged at the top and the cylinder at the bottom so as to allow the necessary play.

Air-lift Finger Chute Gate.—For most of the important ore chutes of the Oliver mines, at Ely, Minn., a modified finger gate with a homemade air lift is used. The drawing, Fig. 222, represents the arrangement for

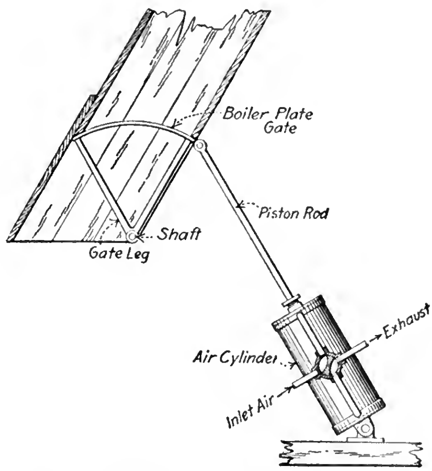


FIG. 221.—GATE OPERATED BY AIR CYLINDER HINGED BELOW.

one of the underground skip-loading chutes. The fingers of the gate are made of rails, the weight depending chiefly on the stock of material which it is desired to use up. In some cases, instead of bending the rails, they are made in two pieces and riveted together. The lift cylinder is made of a piece of 4-in. pipe. One end of this is closed with an ordinary cap and the lift is hung by this. The other end is closed with a reducer which contains an inlet for the air, and a stuffing box through which the rod runs. The piston forming the end of the rod consists of a leather gasket with turned-down edges, between two metal disks with nuts on each side. The release of the air to drop the gate is effected by the same valve that admits the air. This is a three-way valve made from an ordinary stop-cock. The body and plug are drilled as shown in the detailed drawing. Position A shows the valve open to the compressed-air supply for raising the gate. By turning 90°, B, the drilled hole in the plug, registers with the passage of the cylinder and the part through the plug registers with the hole drilled in the body, thus permitting the air to exhaust.

Underswung Rack-operated Arc Gate (By Walter R. Hodge).—The chute gate shown in Fig. 223 is of the underswung arc type, hung from heavy castings, which are bolted to the timbers of the chutes or bin and not to the chute lip, as they often are. These suspending members are made unusually heavy and extend well out from the face of the timbers

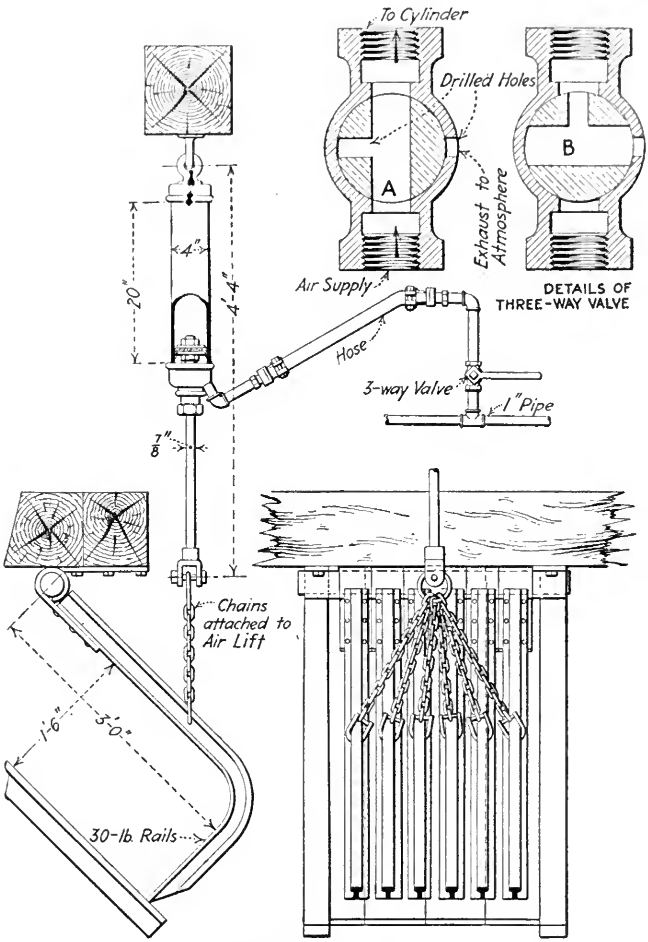


FIG. 222.—ASSEMBLY OF GATE AND SECTION OF VALVE.

to allow the top of the door to clear the bottom of the chute at the extreme open position. The method of operating the gate is perhaps a little unusual. Two racks are fastened by pins to lugs riveted to the lower edge of the door. The racks are inverted, teeth down, and so are protected from dirt or fines. No lubricant is used. Each of these racks works on

a pinion, keyed to a shaft. This shaft lies back and under the chute and turns in gas-pipe bearings set in the timbers. A crank operates the shafting and pinions. One revolution of the crank raises or lowers the door 6 in. and effectually checks any rush of ore. The rock caught on the closing edge of the door, instead of clogging the door, is thrown back into the chute by the rising door or pushed into the car. The whole mechanism is simply and easily operated by one man. Gates of this type are used at the Burra Burra mine of the Tennessee Copper Co. in connection with large pocket chutes from which tram cars are loaded.

Safety Lever for Arc Gate (By E. W. R. Butcher).—In its effort to reduce accidents, the Republic Iron & Steel Co., on the Mesabi range, has introduced among numerous safety devices, an operating lever for the

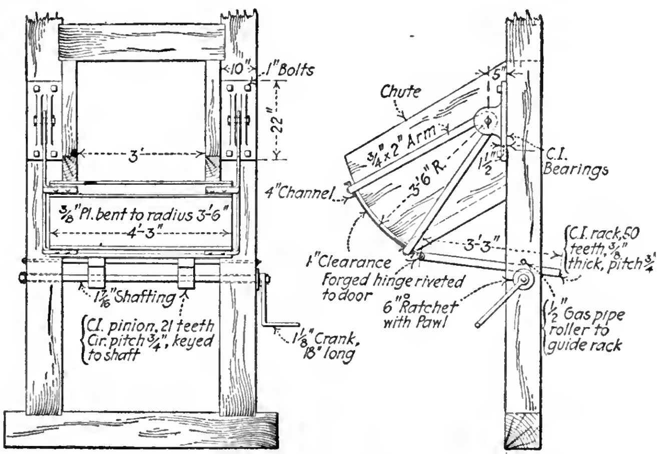


FIG. 223.—ELEVATIONS OF CHUTE AND RACK-OPERATED GATE.

underground arc-type chute gates. Fig. 224 shows its details and arrangement. Usually such a lever arm is attached directly to a trunnion of the gate. The arm extends across the drift and where motors are used, it is necessary for the man emptying the chute to stand between the cars. In this position, he is likely to be caught between the car and the arm should the train move unexpectedly. Furthermore, while, as a rule, the chute-man is required to remove the arm when the chute is emptied, if he fails to do so it is likely to strike anyone riding past on the motor, and to cause a serious accident.

With the device here shown, the lever arm is on the side of the drift, making it unnecessary for the chute-man to stand between the cars and largely doing away with the danger of accident. The ends of the gate arm, of the cap arm and of the double-jaw connecting piece are fitted with several connecting holes by which the arms can be adjusted to the posi-

tions which permit the most satisfactory operation. Placing the connecting end of the gate arm in a horizontal position or at a slight angle above the horizontal and the connecting piece of the cap arm at 45°, gives the best results. The ends of the connecting arms are made round and the holes in them are made a little larger than the holding pins, so as to prevent binding. The measurements given are for a chute 30 in. wide in a drift 7 ft. wide. The main dimensions can be easily changed to suit any ordinary width of chute or drift.

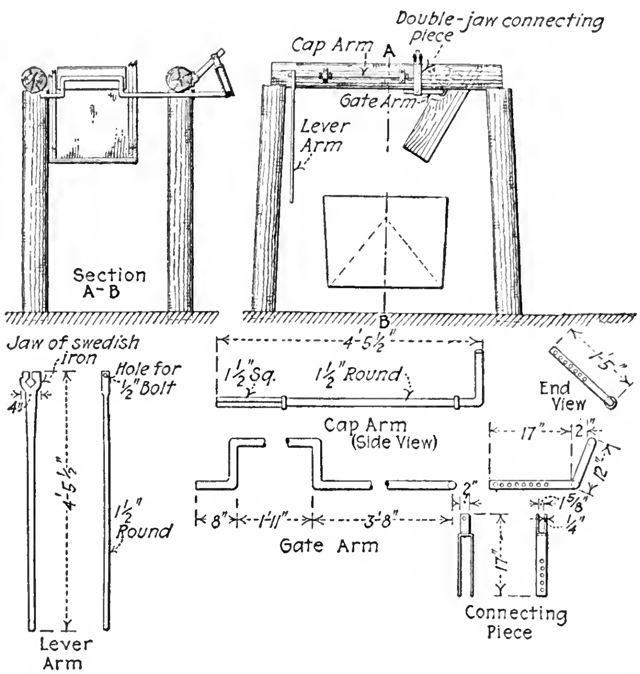


FIG. 224.—ARRANGEMENT FOR SAFELY CONTROLLING CHUTE GATES.

SKIP POCKETS

Hydraulically Operated Skip Pockets (By Clarence M. Haight).—Fig. 225 shows the general arrangement of the steel ore-loading pockets in use in the Palmer shaft of the New Jersey Zinc Co.'s mine at Franklin Furnace, N. J. The pockets are the design of R. M. Catlin, superintendent. The arrangement includes a loading pocket, and between this and the rock ore-bin, a chute and gate. The loading pocket holds one skip-load of ore only; the upper chute feeds the loading pocket and its gate cuts off the ore supply therefrom when the skip is filling. Water from the column pipe in the shaft furnishes power to the hydraulic ram which

operates the gate levers. The direction of the movement of the ram is controlled from the loading platform by an easily operated four-way valve. Water, after use, flows out through a waste pipe from the valve.

When closed, the apparatus stands as shown in the drawing. The lower door is closed and the lip A of the upper chute gate is drawn back beyond the bottom of the chute. This allows the ore to run into the bottom or loading pocket until that is filled, and, as the angle of repose of the ore is then reached, the movement of the ore stops with both the lower pocket and the chute full. When the empty skip is spotted at the door of the loading pocket, the controlling valve is reversed. The hydraulic

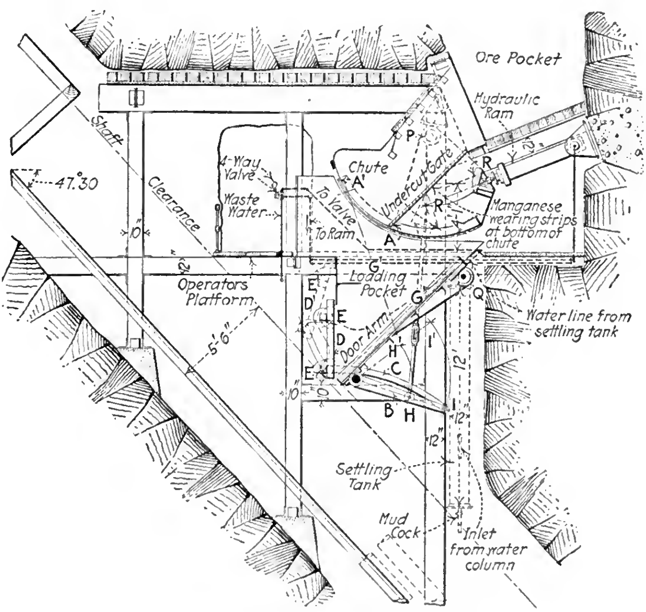


FIG. 225.—SIDE ELEVATION OF SKIP POCKET.

ram moves out along the arc shown, its connection point R moving toward the position R'; this moves the lip A around its pivots P toward A', closing off the chute so that no more ore can go into the loading pocket. Meanwhile, the rod G is pulled forward and up toward the position indicated by the line G'. This motion moves the lever arms B and C, which link the rod G to the lower pocket door, so that the pins H and I advance toward the positions H' and I'. The door D of the loading pocket is held against the mouth of the pocket by two arms, pivoted at Q and E. The movement of the arms B and C forces the door toward the open position D'. This position is reached at the end of the stroke of the hydraulic ram. As this door opens, the ore runs from the pocket

into the skip. In this operation, the advance of the lip *A* is more rapid at first than the movement of the door *D*, so that it has cut off the ore lying in the chute before the loading pocket begins to discharge to the skip. In like manner, on reversing the ram, the door *D* moves the more rapidly at first, so that it is closed tight before the lip *A* has retreated far enough to allow any ore to run into the pocket. Hence, all danger of an excess of ore running over the skip and down the shaft is avoided. The lip *A* cannot jam, because it cuts in from under the chute when closing and so cannot catch on a chunk of ore; while, as the door *D* does not close until the pocket is empty, there is nothing for that to catch on.

One man only is needed to operate a pair of pockets, as the valves work easily. A second man is stationed in the shaft, below the loading platform, to trim the skip in case a large chunk lies on the top of the load

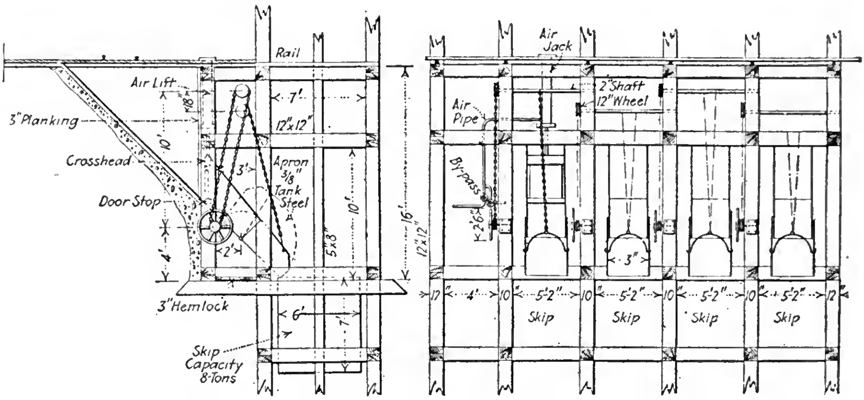


FIG. 226.—BINS AND LOADING CHUTES, HANCOCK NO. 2 SHAFT STATIONS.

and projects over the back or the sides of the skip. The operation of these pockets is easy and rapid. With skips averaging about 6 tons to the load, over 2000 short tons has been hoisted in eight and one-half hours.

Hancock Skip Pockets (By Claude T. Rice).—The details of the construction of the doors to the skip pockets in the Hancock Shaft in the copper district of Michigan are shown in Fig. 226. The chutes are equipped with an apron to bridge the gap between the bin chute and the skip. This apron is raised by a handwheel and a system of chain-driven pulleys. The apron falls by gravity and catches on the edge of the skip, while the skip tender has to raise it by means of the wheel. These wheels can be whirled rapidly, making the pawls sing as they skim over the ratchet and the apron is moved away faster than would at first be expected.

The gate proper of the bin is a corrugated plate operated by an air

cylinder. Air is admitted under the piston only for the raising of the gate, while the lowering is regulated by the opening of another valve which controls the escape of the air from under the piston. The gate plate is carried from a crosshead that moves in the same steel guides as the plate itself. The guides are bolted to the posts of the sets put in to carry the front lining of the bin. At the bottom of the guides are stops to keep the gate from touching the bottom of the discharge chute; if it did there would be a tendency for the gate plate to bind in its guides, owing to the fact that the chute bottom is sloping. A short chain carries the plate from the crosshead, while the crosshead connects directly to the piston of the air cylinder. This cylinder is placed so that the top comes just a little above the floor of the station and is carried by the floor.

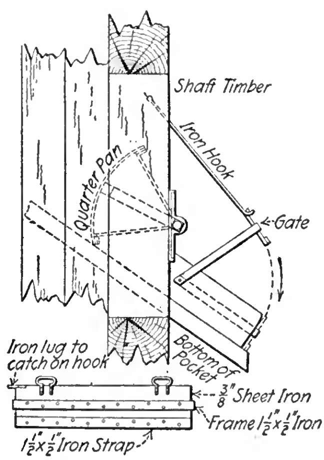


FIG. 227.—EMERGENCY GATE FOR SKIP POCKET.

The reason for corrugating the gate is to diminish the friction of the ore, since the pieces of ore have a tendency to bridge across the corrugations and touch only the high places. The gate opens easily, and its weight, together with the weight of the crosshead and the plunger of the air cylinder, closes it. In order to allow the men to lift without any danger the dogs on the ratchet that controls the descent of the apron, the dogs are provided with handles.

Emergency Ore Gate (By L. D. Davenport).—In one of the mines of the Chisholm district, on the Mesabi range, considerable difficulty was experienced in cutting off the flow of ore from the underground pocket as soon as the skip was full. The emergency gate, shown in Fig. 227, was designed to overcome this trouble and is now in general use in the mines in this district. As shown in the accompanying sketch, the gate consists of a simple 3/8-in. iron plate 8 in. wide and long enough to cover

the end of the chute. It is supported in such a manner that it can be instantly dropped over the end of the chute and thus stop the ore in case an obstruction prevents the quarter pan from closing down tight.

HEADFRAMES

Headframe with Guy-rope Bracing.—The headframe at the No. 4 shaft of the St. Louis Smelting & Refining Co. in southeastern Missouri, is approximately 70 ft. high, and no bracing legs are used in the construction. The frame is surrounded by the shaft house with a landing platform for pulling off the cars at the level of the top of the bins, which are about 30 ft. high, but this shaft house and bin do not aid in steadying

Table I.—Timber List

Number	Mark	Size	Length	Number	Mark	Size	Length
3	A	8× 8 in.	28 ft. 0 in.	2	G^4	6× 8 in.	9 ft. 0 in.
1	A^1	8× 8 in.	20 ft. 0 in.	1	G^5	6× 8 in.	6 ft. 3 in.
8	A^2	8× 8 in.	4 ft. 0 in.	4	H	4× 6 in.	11 ft. 8 in.
6	B	8× 8 in.	20 ft. 0 in.	12	H^1	2× 3 in.	2 ft. 5 in.
8	B^1	8× 8 in.	4 ft. 2 in.	2	H^2	2× 3 in.	1 ft. 0 in.
1	C	8× 8 in.	20 ft. 0 in.	4	H^3	4× 4 in.	3 ft. 0 in.
1	C^1	8× 8 in.	15 ft. 0 in.	1	H^4	4× 4 in.	4 ft. 9 in.
2	C^2	8× 8 in.	16 ft. 0 in.	1	H^5	3× 3 in.	6 ft. 3 in.
2	C^3	8× 8 in.	20 ft. 0 in.	1	H^6	3× 3 in.	11 ft. 8 in.
2	C^4	8× 8 in.	5 ft. 4 in.	1	H^7	3× 3 in.	5 ft. 7 in.
2	C^5	8× 8 in.	6 ft. 8 in.	2	H^8	2× 3 in.	6 ft. 6 in.
2	C^6	8× 8 in.	12 ft. 2 in.	2	H^9	2× 3 in.	6 ft. 3 in.
2	C^7	8× 8 in.	11 ft. 5 in.	20	H^{10}	2× 8 in.	7 ft. 0 in.
3	D	6×10 in.	12 ft. 9 in.	1	H^{11}	3× 3 in.	33 ft. 0 in.
3	D^1	6×10 in.	22 ft. 0 in.	2	I	2×10 in.	36 ft. 4 in.
4	D^2	6×10 in.	4 ft. 4 in.	5	I^1	3× 4 in.	5 ft. 4 in.
4	D^3	6×10 in.	11 ft. 4 in.	1	I^2	2× 3 in.	2 ft. 3 in.
3	E	8× 8 in.	16 ft. 0 in.	1	I^3	2× 3 in.	12 ft. 6 in.
1	E^1	8× 8 in.	9 ft. 8 in.	1	I^4	2× 3 in.	4 ft. 0 in.
3	E^2	8× 8 in.	7 ft. 0 in.	1	I^5	2× 3 in.	11 ft. 2 in.
4	E^3	8×10 in.	4 ft. 2 in.	1	I^6	2× 3 in.	2 ft. 0 in.
3	E^4	8× 8 in.	6 ft. 3 in.	38	I^7	1½×10 in.	1 ft. 7 in.
4	F	6×10 in.	7 ft. 6 in.	1	I^8	6×10 in.	1 ft. 4 in.
3	F^1	5× 8 in.	11 ft. 2 in.	4	J	6× 4 in.	3 ft. 0 in.
5	F^2	5× 8 in.	9 ft. 6 in.	2	J^1	6× 4 in.	2 ft. 6 in.
2	F^3	5× 7 in.	9 ft. 6 in.	20	K	3× 9 in.	10 ft. 2 in.
1	F^4	4× 8 in.	6 ft. 8 in.	14	K^1	3× 9 in.	5 ft. 11 in.
2	G	3×10 in.	6 ft. 0 in.	14	K^2	3× 9 in.	various
2	G^1	3×10 in.	12 ft. 0 in.	13	K^3	3× 9 in.	9 ft. 8 in.
6	G^2	3×10 in.	6 ft. 0 in.	3	L	4× 4 in.	4 ft. 6 in.
3	G^3	8× 8 in.	8 ft. 0 in.	3	L^1	4× 4 in.	10 ft. 6 in.

the structure. The bracing is obtained by four 7/8-in. guy ropes that are anchored to concrete deadmen. The guy ropes are tightened by turnbuckles or else by pulling the rope back on itself after passing around a small sheave anchored to the deadman. This wheel is large enough to permit the rope to be pulled around it without injury, by a block and fall fastened to the end of the rope and attached to the rope itself. This is the best way of tightening the guy rope.

The ore is hoisted on one-deck cages in 1-ton cars, and little trouble is experienced from vibration of the headframe. There are several of these headframes in the district.

A-type Timber Headframe (By G. A. Denny).—The headframe represented in Fig. 228 is one installed at a Mexican mine. It was designed for a shaft inclined at 65°, for a load of 4000 lb. and a maximum rope speed of 460 ft. per minute. The cost of erection in Mexican currency was as given in Table III. Tables I and II give bills of materials for timber and iron.

Table II.—Bolt and Washer List

No.	Mark	Bolts		Length	Washers	
		Type	Size		No.	Angle
1	a	B	1¼ in.	15 ft. 8 in.	2	82°
1	a¹	B	1¼ in.	12 ft. 8 in.	2	82°
2	b	B	1⅛ in.	10 ft. 0 in.	4	flat
2	c	B	1⅛ in.	10 ft. 3 in.	2	flat
					2	65°
1	c¹	B	1⅛ in.	9 ft. 9 in.	1	flat
					1	65°
3	d	B	1 in.	9 ft. 6 in.	6	45°
6	d¹	B	⅞ in.	7 ft. 6 in.	12	flat
3	d²	A	⅞ in.	6 ft. 9 in.	6	flat
6	e	A	¾ in.	1 ft. 10 in.	12	65°
6	e¹	A	¾ in.	2 ft. 5 in.	12	65°
6	e²	A	¾ in.	3 ft. 0 in.	12	65°
21	f	A	¾ in.	1 ft. 5 in.	42	flat
3	f¹	A	¾ in.	1 ft. 7 in.	6	flat
21	g	A	¾ in.	1 ft. 1 in.	....	....
11	g¹	A	¾ in.	1 ft. 8 in.	13	flat
72	g²	A	¾ in.	0 ft. 11 in.	....	....
3	h	B	¾ in.	2 ft. 3 in.	6	flat
18	j	A	¾ in.	0 ft. 11 in.	18	flat
6	k	C	see drawing		3	flat
					3	72°
33	m	D	1 in.	....	66	flat (see drawing)

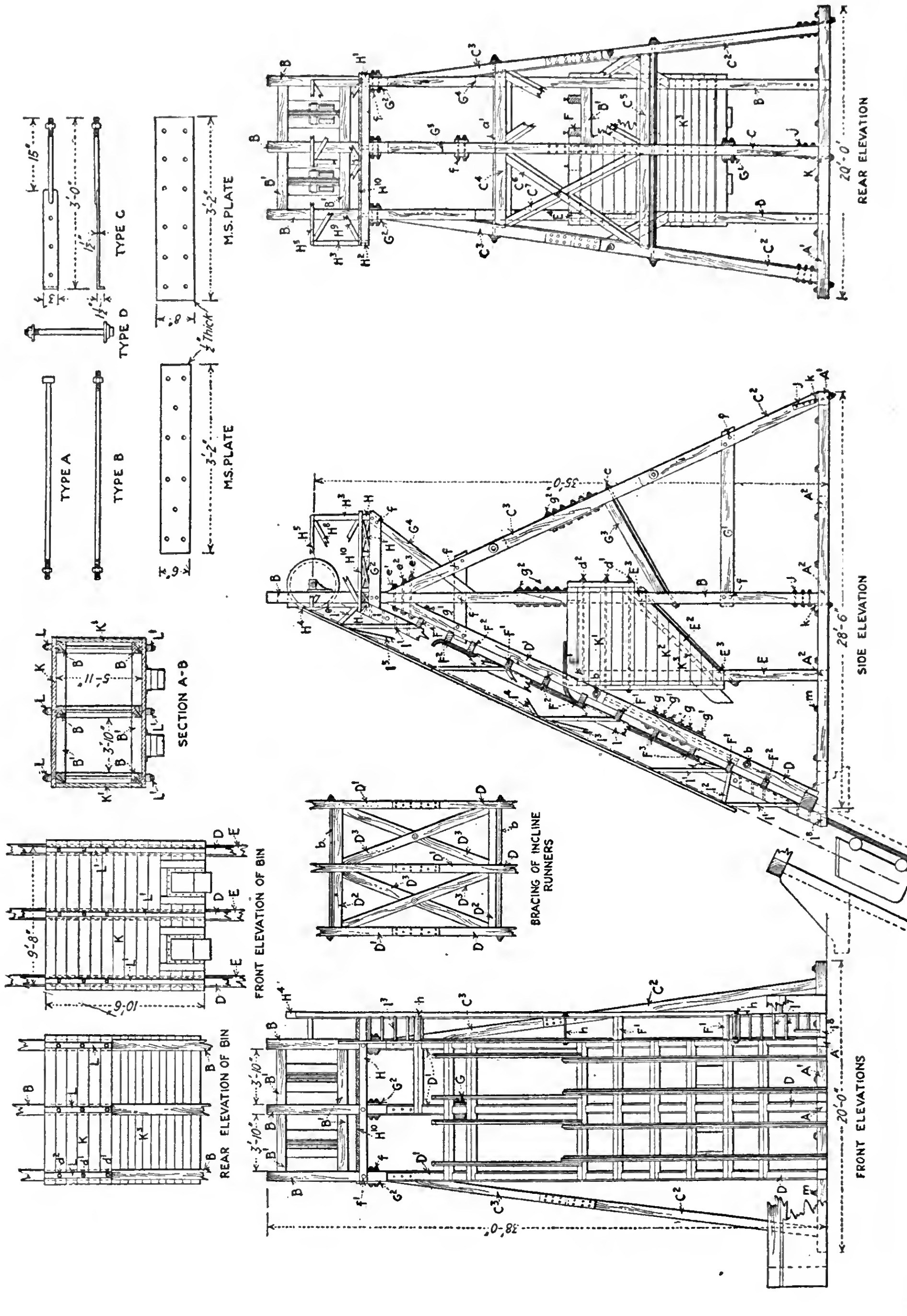


FIG. 228.—ELEVATIONS AND DETAILS OF SMALL TIMBER HEADFRAME FOR INCLINED SHAFT.

TABLE III.—COSTS

	Pesos
Timber (at 50 pesos per M)	230
Carpenters (at 3 pesos per day)	772
Smiths (at 2.50 pesos per day)	92
Mechanics (at 3 pesos per day)	65
Iron plates and bolts	293
Concrete foundations	113
Laborers	70
Freight	35
Administration	375
Total cost	2045

Concrete Headframe with Fleeting Device (By L. O. Kellogg).—The flat-lying shaft of the Sterling Iron & Railway Co. in southeastern New York reaches the surface at an angle only a little greater than the slope of the ground. The hoist is set on the hill about 100 ft. above the collar and the dump, bin and headframe, built in this space, at no place rise more than 12 or 15 ft. above the ground. There is thus no use for a high headframe and all that is used is a concrete foundation under the sheaves and some channels to support the track. The general arrangement is shown in two elevations in Fig. 229. The 10-in. channels extend toward the shaft collar and carry crossties on which the rails rest. The centers of the sheaves and of the drums being so close together and the 2500 ft. of rope requiring several wraps on the 3 × 6-ft. drums, it is evident that smooth winding could not be had without some provision for taking care of the fleet. For this reason, the sheaves are made to revolve on the shaft which is held fixed, contrary to the more usual practice of keying the sheave to the shaft and allowing the latter to revolve in bearings. The sheave is consequently free to move laterally and thus follow the rope in its travel across the face of the drum. The correspondence is not perfect, the 10-in. hub of the sheave reducing its possible lateral travel to 28 or 30 in. while the rope travels 36 in. across the drum. In practice, however, the rope winds perfectly. The sheave hub is lined with brasses about 1/4 in. thick and lubrication is had by means of two grease cups in the middle of the hub, diametrically opposite. The shaft is 5 in. in diameter and is held in the castings which rest on the concrete piers, by means of setscrews.

Derrick for Sinking 557 Feet.—Shaft No. 6 of the St. Louis Smelting & Refining Co. in southeastern Missouri was sunk to a depth of 557 ft. with a boom derrick. The derrick is rather generally used in Missouri and Michigan for starting sinking operations, but 100 ft. has usually been the limit of depth to which it is applied. Considerations of cost and convenience led to its adoption in this instance.

The derrick was set opposite the center of the long side of the shaft and 38 ft. back from the center line. It was steadied by six guy lines. The mast was 40 ft. in height and the boom 44 ft. An elevation of the derrick is shown in Fig. 230, and a plan of the whole layout. The bottom of the mast was carried by the footstep casting shown, to which it was

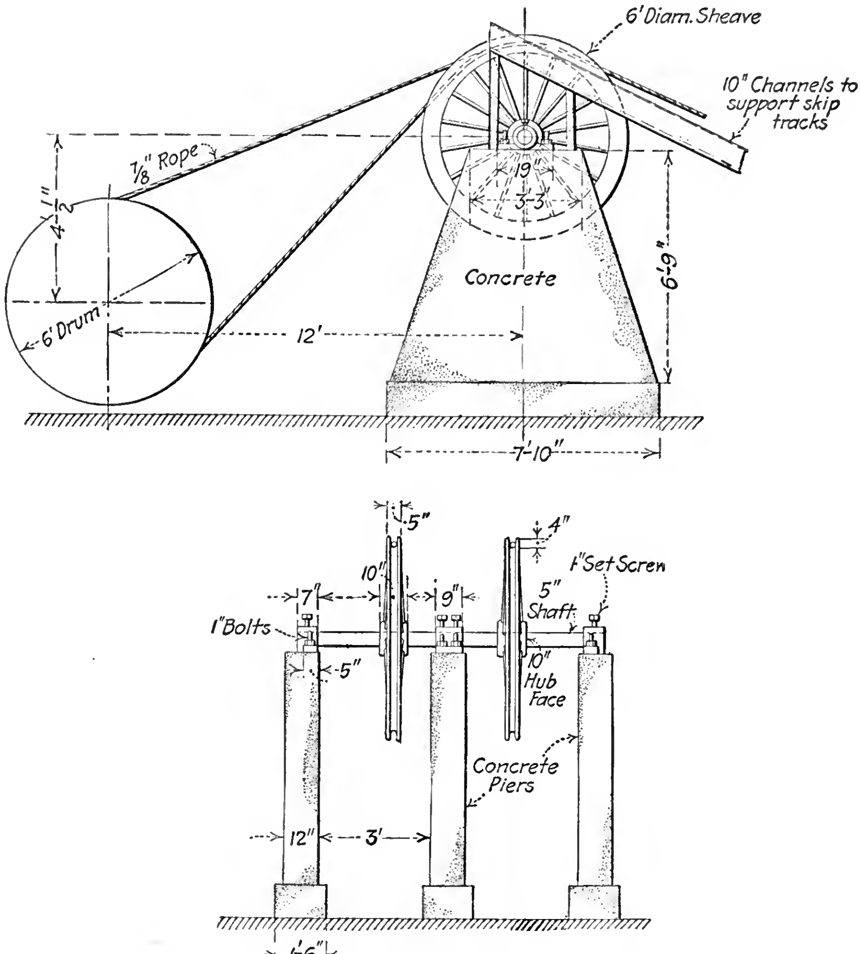


FIG. 229.—HOISTING SHEAVES REVOLVING ON FIXED SHAFT.

bolted. This footstep rotated in a bearing block made of 10 × 10-in. timbers bolted together as illustrated, with a hole in the center fitted to the footstep shaft. The top of the block carried a 4½-ft. square plate around the center bearing hole. The block was bolted to the frame structure supporting the derrick. Rotation of the derrick was ob-

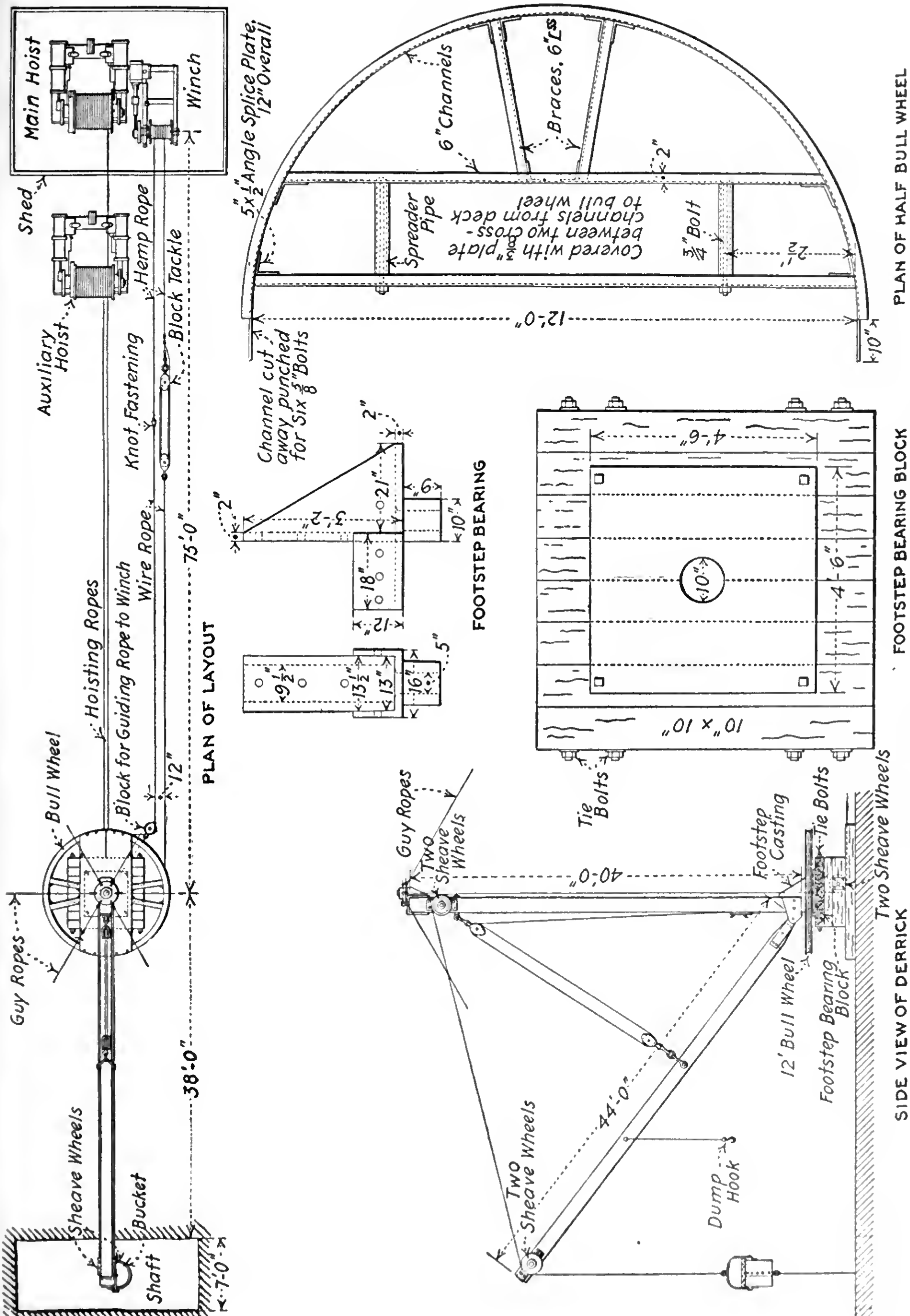


FIG. 230.—LAYOUT AND DETAILS OF SINKING DERRICK.

tained through the 12-ft. bullwheel, the construction of which is also shown in detail. It was built of 6-in. channels, one forming the circumference and the others acting as braces. The wheel was attached to the cast footstep. The circumferential channel was turned with the ribs out to form a trough for the actuating rope.

Two hoists and a winch were employed in conjunction with the derrick. The winch was used for rotating the bullwheel. A small wire rope was wrapped several times around the bullwheel and one end brought back as a tangent to a point near the winch. The other end was brought around to a point near the first end and turned in the same direction by a block and also brought near the winch. On the 12-in. winch drum a hemp rope was used, to avoid bending a wire rope to so small a radius. This rope had its ends attached to the ends of the wire rope, one attachment being brought through a block and tackle for taking up slack.

The main hoisting cable passed over a sheave on the end of the boom and another on top of the mast, down through a 5-in. hole in the footstep casting, over another sheave attached to the supporting framework and thus to the hoist. By this arrangement the swinging of the derrick did not interfere with the operation of the rope. The rope to the auxiliary hoist followed through a similar series of sheaves placed on the opposite side of the boom and mast. The slope of the boom was changed by hand when necessary, using a block and tackle from the top of the mast to a point near the center of the boom. But the bucket usually operated in the shaft center and for that reason the slope of the boom was seldom changed. The arrangement of the hoists is shown in the plan. The winch was near the main hoist so that the engineer had easy control over both; each hoisting operation of course necessitated a turning of the derrick. Usually it was not necessary to operate both the auxiliary and main hoists at the same time, but when it was, a top-man could manipulate the auxiliary, as it was used only for material. The main hoist had a 4-ft. drum and 15 $\times$ 18-in. cylinders. The auxiliary had a 30-in. drum and 8 $\times$ 10-in. cylinders. The winch had a 4 $\times$ 4-in. engine.

The bucket was unguided in the shaft, but no difficulty was experienced from spinning. It was dumped in the usual manner by the rope and hook shown. An unusual piece of equipment was a platform on which the men worked while timbering, placing pipe or barring down the sides. This so-called "butterfly" was handled by the main hoist, being attached by four ⅞-in. ropes from the main timbers of the framework. It was a great convenience, giving opportunity for spreading out the tools while working and being capable of swinging to any desired corner of the shaft.

Advantages attendant on using a derrick are: (1) All surface work is done away with near the collar of the shaft and the danger of objects falling on the men at the bottom is reduced. The men themselves were unloaded at least 35 ft. from the collar. (2) There is no structure near the shaft collar to be injured by flying rock while blasting the upper sections of the shaft. (3) The derrick is cheap to erect and on the completion of any one job is available for another. (4) By the use of two hoists, one for men and one for material, the work of pipe and timber erection is greatly facilitated. (5) By rotating, a great deal of lifting is avoided both in the shaft and around the collar on the surface.

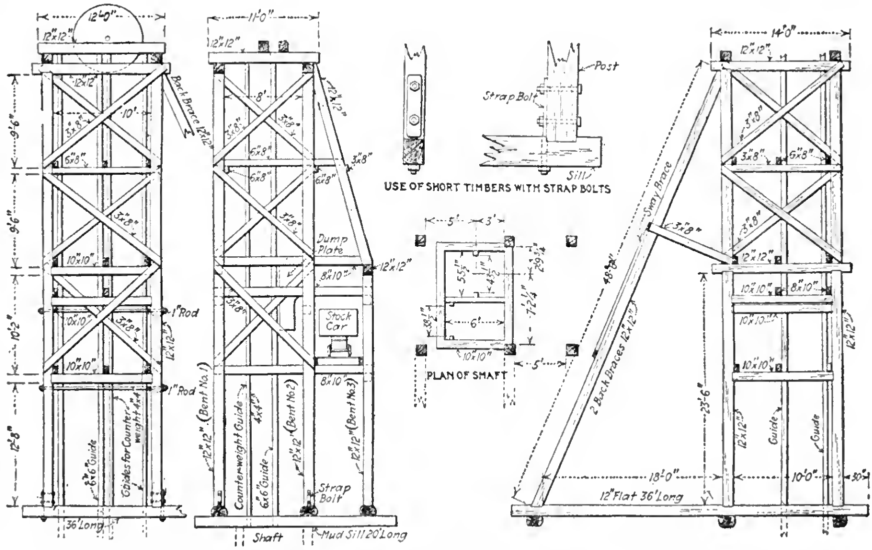


FIG. 231.—ELEVATION AND DETAILS OF SMALL FOUR-POST TIMBER HEADFRAME.

Small Four-post Headframe (By H. L. Botsford).—Fig. 231 illustrates a small timber headframe. It was designed for use at an exploratory shaft, from which it was desired to ship ore from time to time as the work of development progressed. Where long timbers are not obtainable or are too expensive, the headframe may be built in sections, with intermediate caps, and holding-down bolts between the sections. Strap-bolts answer this purpose and cost less than long rods running the full length of the section, from cap to sill.

Reversible Temporary Headframe (By P. V. Burgett).—Fig. 232 shows a round-timber headframe suitable for shaft sinking and development work. It is very simple in construction, and can be quickly and cheaply erected. As first designed, it consisted of only the middle bent *A*, the "rakers" *B*, the dumping platform *D*, and a 4-ft. sheave wheel

E. The permanent hoisting engine, on the same side of the shaft as the "rakers," was to be used for hoisting. Later it was found that the hoisting plant could not be built in time. So the back braces *C* and a 2-ft. sheave wheel *F* were added to the headframe, and a temporary hoisting engine and small upright boiler were erected on the opposite side of the shaft from the permanent engine. The details of the middle bent *A* are shown in the drawing. The back braces and "rakers" have no cross-pieces for braces except those used to support the sheave wheel. The shaft has three compartments; two 5-ft. skipways and a 3-ft. ladderway. During shaft sinking the middle compartment was used for hoisting.

Tripod Headframe (By G. E. Le Veque).—A serviceable tripod headframe is made of three 32-ft. poles, having 12-in. butts. These are bolted at the top, as shown in Fig. 233, and a sheave hung from the 1-in. bolt. The bottoms of the poles are spread to form a 30-ft. equilateral triangle. The hoisting compartment is continued as a 6 × 7-ft. inclosure by 6 × 6-

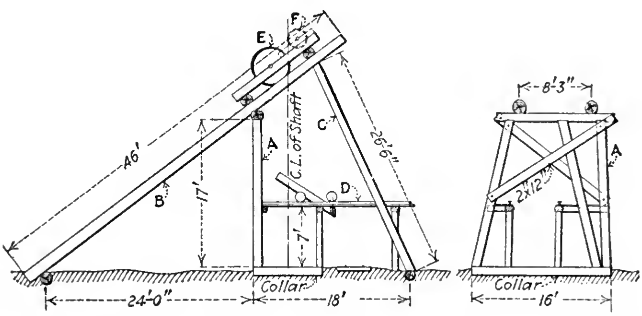


FIG. 232.—PROSPECTING HEADFRAME FOR HOIST ON EITHER SIDE.

in. posts braced by 2 × 6-in. stringers, spiked to the tripod legs 9 ft. below the sheave. Eight ft. above the shaft collar a landing and chute are built in, a space 4 × 4-ft. being left for the bucket and a platform built at one side for the lander. The dumping device consists of a chain with a hook in the free end, fastened to a stringer directly over the chute. In operating, the bucket is hoisted to position *C*, the chain is hooked in the bail ring and the cable slacked, transferring the weight of the bucket from the cable to the chain and bringing the bucket directly over the chute, where it can be dumped by overturning to position *B*. The length of the chain should be adjusted to allow the bucket to rest lightly on the chute when in dumping position and prevent its turning entirely over and dropping material into the shaft.

Prospecting Headframe with Automatic Dump (By Charles Mentzel).—A little consideration of the stresses in a headframe and a few minutes calculation will show that a frame of good design and sufficient strength and stability for sinking purposes can be constructed at considerably

less cost than the four-post frame, cross-braced on all sides, so often found over a prospecting shaft. In addition, the greater convenience, decrease in surface attendance and greater speed in handling buckets, owing to the self-dumping arrangement, give the type illustrated in Fig. 234 a decided advantage. This headframe can be used for either vertical or inclined shafts. In the former, skids are easier to place in the shaft than are guides, and they obviate the use of the crosshead, often a source of danger. Where the ground is good, all the timber necessary is a wall plate about every 8 or 10 ft. tightly wedged to the ends of the shaft. The skids are spiked to the plates. Prospecting shafts usually follow the ore and,

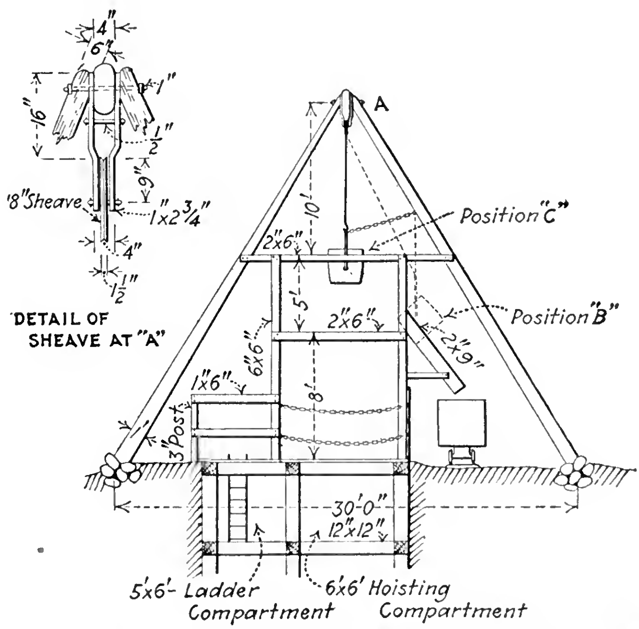


FIG. 233.—TRIPOD PROSPECT HEADFRAME AND LANDING PLATFORM.

therefore, are almost always inclined; and generally the inclination varies from time to time.

The essential points entering into the design of a headframe are the height of the car used to remove the hoisted muck, and the distance of the hoist from the shaft. If it is considered advisable to build a frame with a pocket, which will do away with the services of a top man on night shift, it is calculated to hold about 30 tons and to fit between the front posts.

The construction of the bucket is the next consideration. A bucket of approximately 10 cu. ft. capacity is usually about 26 in. diameter at the top, 24 in. at the bottom and about 34 in. deep. Two horns, 8 or 9 in. long and 1¼ in. in diameter, are fastened to opposite sides of the

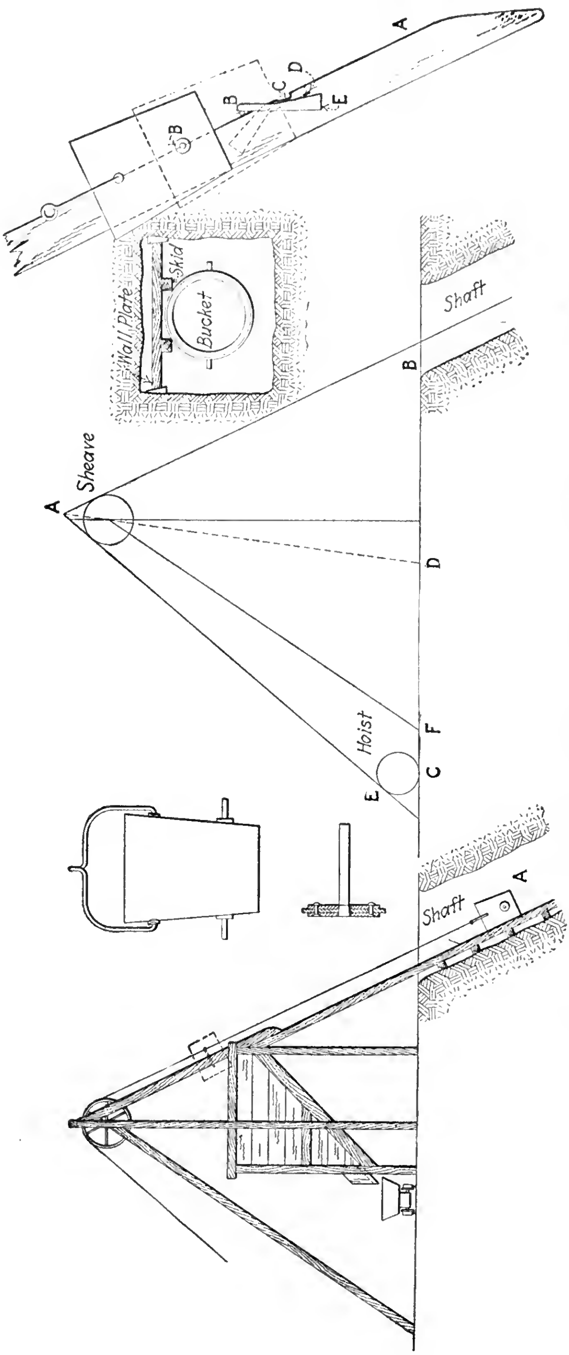


FIG. 234.—DETAILS OF HEADFRAME, BUCKET AND AUTOMATIC BUCKET DUMP.

bucket about 9 or 10 in. from the bottom, as shown. They are attached to flanges riveted inside and outside the bucket as shown in detail below the view of the bucket itself.

The position of the sheave depends upon the height of the dumping device and the amount of overwind allowable. In prospecting shafts the danger of overwinding is small with the slow, low-power, geared hoists usually employed, so 5 ft. is sufficient, and taking the length of bale, clevis, thimble and about 2 ft. of cable doubled to clamp it, gives about 10 or 12 ft. as the distance from the dump to the tangent point of the sheave wheel. Knowing the diameter of the sheave and the size of the pillow blocks, the vertical posts can be readily located. Having the front posts, the back braces may be drawn as follows: Scale off the distance of the hoist from the collar as shown in the diagram. Draw *AE* tangent to hoist drum and sheave. Theoretically the line *AD* bisecting the angle *EAB* is in the best position to receive the pull of the hoist and the resisting pull of the load. *AD*, however, may be too close to the line of the vertical posts to ensure stability, and it is better in any event to place it so that the foot comes closer to the hoist at *F*. The positions of the posts and back braces have now been decided. To find out how to place these with respect to the shaft we must consider how the bucket rides on the skids. This is shown in the plan of the shaft, and in the elevation of the headframe and at *A*. The skids are 6- or 8-in. round skinned poles dressed at the parts where they are fastened to the wall plates and are placed so that the bucket rides on them as shown, without touching the wall plates. The horns do not come into contact with anything in the shaft. With this sketch made we have the position of the hoisting cable with respect to the shaft and headframe, and can locate the posts and back braces.

The posts are tied with three girts and a cap, and the back braces with two girts. The batter may be 1:8 or 1:10, depending on the size of the shaft. For a foundation, ordinary log cribwork is built up and filled with waste rock and the stringers spiked to the cribbing with drift bolts. In one frame 10 × 10-in. hewed timber was used and 1-in. tie rods and bolts, and 8 × 8-in. timber for the bin. A smaller frame was constructed of 8 × 8-in. timber and ¾-in. tie rods. The front posts are set at the inclination of the shaft and are connected with the back braces by means of girts and tie rods.

The shaft skids run from the shaft to the point of dump. The dumping skids of 8 × 8-in. timber begin about 6 ft. below the dump and are bolted to the headframe. As the bucket is hoisted out of the shaft, the horns ride on the runners on the dumping skids, which are placed so that the distance between the inner faces is an inch or two greater than the width of the bucket, to allow clearance. The shaft skids prevent

the bucket from tipping on the horns at this point. The bucket continues up the dumping skids until the horns drop into a dap on each skid. At this point the hoist is stopped and the bucket allowed to swing by gravity, thus dumping. The bucket is now raised a foot or so and then lowered. In lowering, the horns engage trippers which are thrown over the daps, allowing the horns to slide down without catching in the daps. The trippers then swing back by gravity leaving the daps free to hold the horns of the bucket on the next trip.

The dumping device is simple and can be made by any mine blacksmith. The construction is shown on the right of Fig. 234. The tripper *E* is shown in full lines in its normal position. When the horn *B* engages it on its down trip it carries it with it until it is stopped by the catch *D*. This enables the horn to go past the dap *C* without being stopped by it, as shown by the dotted lines. After the bucket has passed, the tripper swings back to its first position by gravity, since the lower part of the

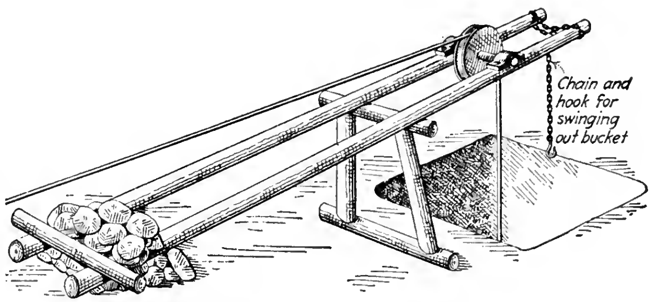


FIG. 235.—HOISTING ARRANGEMENT FOR PROSPECT SHAFT.

tripper is made heavier than the upper. The runner *A* with the dap *C*, which should be at least 2 in. deep, is made of $\frac{3}{8} \times$ 2-in. iron bolted flush with the timber. The tripper is best made of $\frac{5}{8} \times$ 3-in. iron and about 20 in. long and should swing from a 1-in. bolt. The construction is apparent. A headframe of this description can be built at a cost of about $50 for a small one and $150 for a larger one, including all material and labor. Any carpenter of ordinary ingenuity should have no trouble in designing and framing it. For vertical shafts the same form can be used, carrying the skids up and bending them at the collar to meet the frame at the dump. A few saw cuts on the inside of a green pole make it possible to bend it through a considerable angle without breaking. Of course, a roller for the cable is required at the shaft collar in this case.

Substitute for Small Headframe (By Walter R. Hodge).—The type of head rigging shown in Fig. 235 has been used in many places on the Mesabi range in sinking test pits and shallow timber shafts. It would be useful in most places where a temporary headframe is needed for

shallow depths. It consists of two pieces of round timber about 28 ft. long, supported a little more than half way toward the small end by a roughly made bent slightly inclined. The large end is weighted with waste rock or timber. The two timbers are set close together and the sheave revolves between them. The sheave is retained in place by two collars or simply revolves between two heavy pegs driven in the logs. Such a device may do good work to a depth of 200 ft. A small "puffer" is usually the source of power on the Mesabi. A windlass may be framed to the butts of the timbers and leave the collar of the shaft free from obstructions.

TURN SHEAVES

Turn-sheave Types (By Floyd L. Burr).—In any turn-sheave frame whatever, the sheave wheel will be supported by a pair of members whose axes lie in planes parallel to the plane of the sheave. Generally also the plane containing these two axes will be at right angles to the plane of the sheave. These members will usually at their ends frame into main supports which are vertical, horizontal or at right angles to the sheave supports. Four types of turn-sheave or angle-sheave frame may be recognized.

Type No. 1.—When the line of the resultant of the two ropes is steep and either upward or downward, the principal part of the frame may take the form of a pair of members parallel to the resultant reaching upward into the air from a concrete or other base. The sheave is attached to these members and the stress, tension or compression depending upon the direction of the resultant, is transmitted to the concrete base, which by virtue of its weight resists the lifting tendency or by its stiffness spreads the compression over a sufficient area of the soil beneath, while by end bearing and skin friction it resists the tendency to slide horizontally. The tension or compression members may be of steel or wood.

Type No. 2.—When the line of the resultant is comparatively flat, the frame may consist essentially of a pair of strut-beam members parallel to the resultant, framed between two piers or towers, one or both of which will tend to overturn in a longitudinal direction when the rope is stressed. The piers or towers may be of steel or wood or concrete or reinforced concrete, while the strut-beam may be of wood, steel or reinforced concrete. The towers or piers may, as a variation in design, get some of their stability from diagonal tension or compression braces reaching to the ground or to auxiliary piers or natural anchorages.

Type No. 3.—The same sort of a structure as that mentioned for type 2 may be set with the supporting-beam members normal to the resultant. In this case the resultant load subjects the members to the bending stress only, and tends to overturn the towers in a transverse direction.

Type No. 4.—When it is not convenient to set the supporting members either parallel or normal to the line of the resultant, these members may be set in a convenient direction and supported by two towers or piers. The stress effects are of course a combination of those in the preceding two types.

It should also be mentioned that in the last three types there may be only one heavy pier or tower, from which beams may extend as cantilevers to support the sheave wheel. Of course, many variations are possible.

Fig. 236 shows sketches of these various types.

An installation at "C" shaft of the West Vulcan mine, of the Penn Iron Mining Co., Vulcan, Mich., gives an illustration of types 1 and 2. The hoisting ropes emerging from the hoist house travel about 80 yd. toward the east, thence a similar distance to the north to a point near

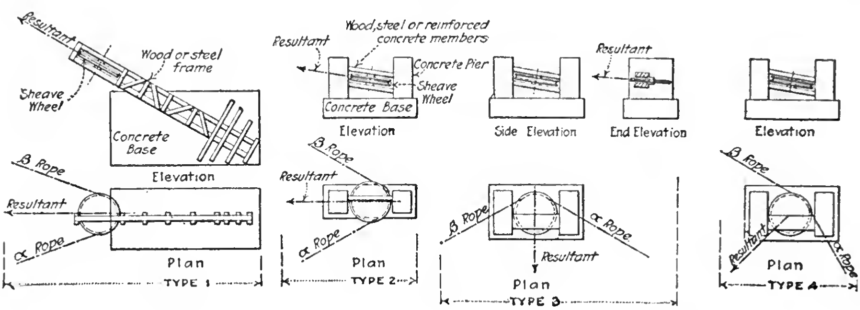


FIG. 236.—THE FOUR TYPES OF ANGLE-SHEAVE FRAMES.

the base of the headframe and thence in a westerly direction to the headsheaves. In their travel to the east and north, the ropes run extremely flat, while in rising to the west toward the headsheaves they are steep. The conditions thus called for a type-2 frame for the first turn, that nearest the hoist, and a type-1 frame for the second. These new frames were to replace old decayed wooden structures and had to be built and made ready for use before disturbing the old ones. It was also necessary so to arrange that there should be no interference with the run of the ropes into the old structures and in the case of the type-2 new structure these ropes had to run through the frame—indeed, through holes cored in one of the concrete piers. The general relations are shown in plan and profile in Fig. 237.

Type-1 structure consists of a concrete base with its top flush with the ground surface, from which emerge three steel frames on a slant. Each frame is to support a sheave, one each for the cage, the skip, and the skip balance. A separate structure will be built in case the future demands a cage balance. Each frame consists of a pair of steel members cross-braced together rigidly below the open space occupied by the sheave and

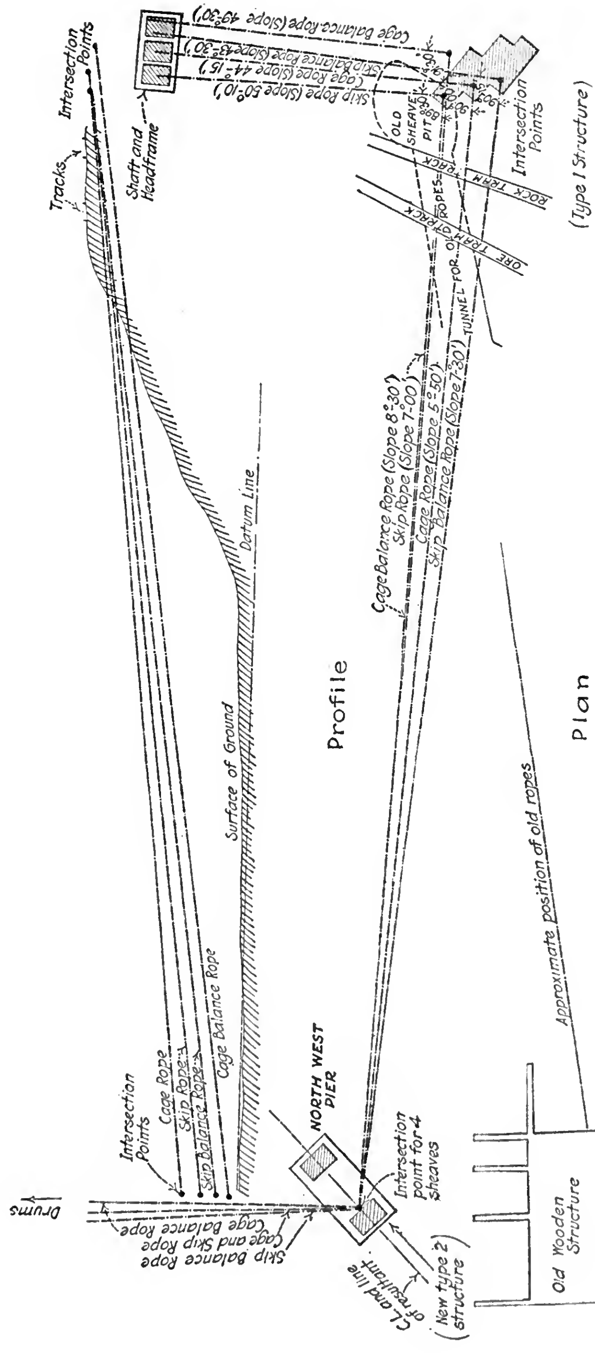


FIG. 237.—LAYOUT OF ANGLE-SHEAVES AT "C" SHAFT, WEST VULCAN MINE.

provided with crosspieces at the lower end for anchorage into the concrete. The concrete base slab is about 7 ft. deep; the steel members reach nearly to the bottom and extend above the concrete to a point just above the bearings of the sheave wheel. The balance-rope sheave will be 6 ft. in diameter; the others are 10 ft. The intersection points were placed about flush with the ground level. Each member is made up of an I-beam with a channel riveted to its upper flange, giving stiffness in two planes. The bearings for the sheave rest upon the upper flange, or channel web, and are attached by bolts which reach through the bottom flange. Angle-steel braces are attached to these members near their upper ends and reaching down to auxiliary anchorage piers help to stay the main members against the bending stresses which occur by reason of the eccentricity of the position of the center of the sheave. This placing of the bearings on one flange of the members is not a thoroughly satisfactory detail and in future installations an attempt at improvement will be made. The center of the sheave and the resultant should coincide with the plane of the center lines of the members, and thus do away with practically all bending moment.

The concrete base being all below the ground level, the placing of the concrete was exceedingly simple. The mixer was set up at one edge of the excavation and after the steel members had been placed in proper position the concrete was merely poured direct from the mixer into the forms—the excavation was in an old rock fill and would not stand without form work. About 75 cu. yd. of concrete was consumed. Considerable irregular reinforcement was placed in the concrete with the idea of binding it together to act as one mass and suspend it from the steel-work. This reinforcement consisted of old steel rope 1⅛ in. or 1¼ in. in size. A force of nine men spent one and one-half days at the work of actual concreting.

The ropes approaching these sheave wheels from a lower elevation had to run under the surface of the ground for about 35 ft. from their intersection with the slope of the hill. The elevation of the intersection points was purposely fixed so that the ropes would enter below the surface in order to pass under the rock and ore tramway tracks. It was not practicable to build concrete tunnels or conduits for the ropes on account of their position about 15 ft. above and directly over an old tunnel through which the old ropes entered the deep pit containing the old angle sheaves. Therefore it was decided to let the ropes run each through a sleeve or conduit consisting of a length of old discarded 36-in. steel stack. These pieces of stack were suspended from stringers spanning the old tunnel and uncertain ground. After the ropes were put into operation in the new sheaves, the old pit and tunnel were filled with waste mine-rock up to and over the conduits. The extreme depth of

the ropes below the surface is about 4 or 5 ft. at the edge of the bank and of course at the point of entering the sheave is nothing.

Type-2 structure shown in Fig. 238*a* consists of a concrete base 12 ft. wide, 25 ft. long, and 5 ft. deep, surmounted by two concrete piers 16 ft. high by 8 ft. wide by 5 ft. long. The top of the base is flush with the general ground surface. The piers are placed symmetrically with the center lines of the base and there is 12½ ft. clear space between them. The long center line coincides approximately with the lines of the resultants and lies in a northwest-southeast direction. Thus there are the "northwest pier" and the "southeast pier." The sheaves, which are 10 and 6 ft. in diameter, are supported by pairs of steel members located one above another and spanning the distance between the two piers. Provision has been made for four sheaves though there are now only the two main sheaves installed. The supports for the 6-ft. sheaves are only partially installed and will be completed whenever needed. Tension in the ropes produces compression in the steel members which push against the northwest pier, tending to bend and break it, to tear it off from the base and to tip the whole structure, base and all, over toward the northwest.

The situation of this structure bears such a relation to the mine workings below that there is a possibility of some settlement. For this reason it was considered necessary to build so that the steel members could upon occasion be removed, rearranged, and put back, in case settlement should occur so as to affect the sheaves more than could be adjusted for by blocking up their bearings. To accomplish this purpose, the steel members are not embedded into the two piers but merely butt against them. They are attached to the piers in such a way as to have their dead weight and that of the sheaves supported. A small space is allowed for the expansion of the steel so as not to put a compressive stress therein nor to put an additional overturning moment or spreading force on the piers.

The northwest pier is calculated as a cantilever from the base slab to resist safely the stressing effects of the breaking of the upper rope while the other three ropes are carrying their working load. The hoisting ropes in use are 1⅛ in., but 1¼-in. ropes were assumed in the computations. The balance ropes were assumed to be ⅞ in. The base slab was reinforced top and bottom to insure its acting as one piece. The northwest pier was reinforced as a cantilever, the reinforcement being continuous with that in the base. This reinforcement consists of discarded 1⅛-in. and 1¼-in. steel ropes. These ropes were used rather lavishly. It was found convenient to let them run up into the southeast pier much as in the northwest pier, so that it is actually reinforced, though there is no definite requirement for such reinforcement. The function

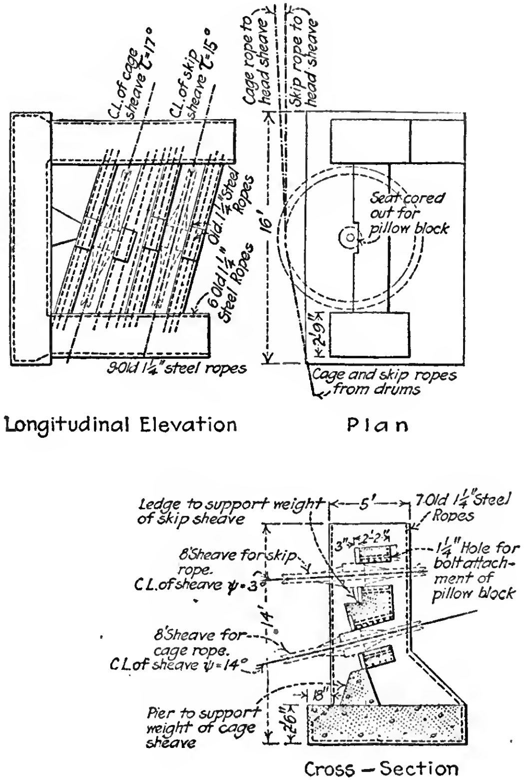


FIG. 238.—CONCRETE ANGLE-SHEAVE FRAME AT EAST VULCAN MINE.

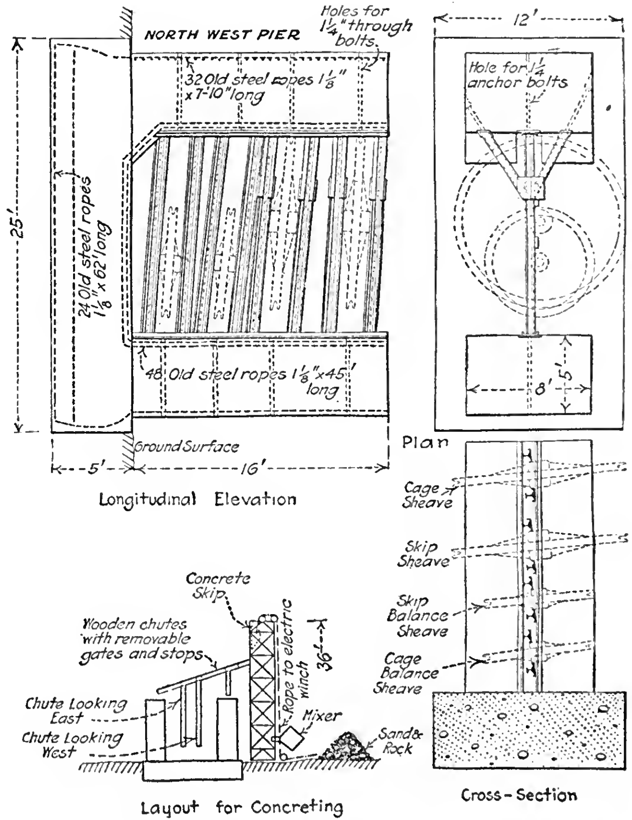


FIG. 238a.—CONCRETE AND STEEL ANGLE-SHEAVE FRAME AT WEST VULCAN MINE.

of this southeast pier is to furnish weight for stability of the whole structure and to support the dead weight of the steel frames and their sheaves. It also serves to resist its share of any incidental lateral stresses.

The pit for the base having been excavated and forms for the piers erected, a number of long ropes were laid in a longitudinal direction in the bottom, then about 18 in. of concrete was placed, embedding the ropes about 2 or 3 in. from the bottom. Also at each end, there were embedded certain anchorages made of old rails. These supported horizontal rails at the inside points of junction between the piers and the base. They were to serve a little later to take the strain arising from tightening the inside face ropes. The ends of the long ropes, the middle portions of which were now embedded in the 18 in. of concrete, were picked up and some of them drawn up tight into the pier forms near their outside faces. Their upper ends were wired to strong wooden beams which had been supported by bents over the forms. The rest of these ends were laid back on top of the 18 in. of the concrete, to be embedded into the bottom of the remaining 3½ ft. of base. It should be mentioned that in casting this 18 in. of base, scrap rails and rods were embedded vertically here and there to bond to it the remaining portion of the base.

The other series of ropes was now put in place. These followed down the inside face of one pier form, then behind the anchorage bar mentioned above, thence horizontally across the intervening 12½ ft. to the other pier at an elevation a few inches below the top surface of the base slab, thence behind similar anchorage bars and up into the other pier form near its inner face. This latter series is of the greatest importance, since it provides the cantilever reinforcement of the northwest pier. The ends of these ropes were wired to overhead wooden beams after the ropes had been well tightened by use of the tackle. Concrete was then poured to complete the base and the piers. The pier forms were of rough planks but were lined with tarred felt both as a nonfreezing precaution and to make the forms watertight. Concrete was mixed in proportions varying from 1 : 2 : 4 to 1 : 3 : 5.

The concreting plant consisted of a No. 11 Cube mixer with loading skip, stockpiles of sand and rock on the ground in the immediate vicinity, a 36-ft. wooden elevator tower containing a self-dumping bucket into which the mixer would discharge directly, an inclined wooden trough or chute with branches above the forms, and an electric hoist for operating the bucket elevator. This plant worked smoothly. A force of 10 men was required to mix and place the concrete. One full and two fractional days were consumed in the concreting. The concrete amounted to 104 cu. yd.

An example of a type-4 structure is furnished by a design now in

process of erection at East Vulcan No. 4 shaft, shown in Fig. 238. This is to carry two 8-ft. sheaves for the cage and skip ropes. The horizontal angle between the ropes lacking only 8° or 10° of 180°, the intensity of the resultant stress is comparatively slight. The structure is designed to be stable and strong, to withstand the effect of breaking one 1¼-in. rope while the other rope is carrying its working load.

This frame consists of a concrete base 2 ft. 6 in. thick and 10 × 16 ft. horizontal dimensions, surmounted at each end by piers 2 ft. 9 in. long and 5 ft. wide. The north pier is 11 ft. 6 in. high, while the south pier is 10 ft. high. Framed between the piers and cast integrally with them are reinforced-concrete beams which are to support the weight of the sheaves (a vertical load) and resist the lateral force of the rope-stress resultant. The resultant tends to bend them toward the east and through the medium of these beams to overturn the piers and indeed the whole structure in the same direction. There is also a small end thrust exerted. Old steel ropes are placed as reinforcement to cause the base to act as one piece and to bind the piers firmly to it. The concrete is a rubble mixture 1:3:5. About 31 cu. yd. is required.

Certain conditions not necessary to mention, fixed the position of the structure and limited the width of the piers. The structure is set with its transverse center line at angles of some 5° or 10° from the lines of resultant of the cage and the skip ropes. For the exact setting of the beam forms, a mechanical scheme was used. This worked nicely for this case but would not be convenient for use in general.

The intersection points, which are on the west side of the structure, were located and marked in space by the heads of nails driven into suitable wooden falsework. It was possible to see easily from these points to points where the ropes would enter the headsheaves and to the drums. Other nails were sighted in along these lines at distances of 15 ft. or so from the intersection points. A suitable wire was stretched along these lines, which were, of course, the future paths of the ropes. Then on the east side of the structure, the transit was "jiggled" into line between the same points at headsheave and drum and lines were stretched to correspond. Thus for each sheave wheel there were had three lines which determined the plane of the sheave. Now by sighting across these lines from east to west, strips were accurately located and aligned on the pier forms for the support of the beam forms. The longitudinal slope of the beams is about 17° for the cage sheave and 15° for the skip, while the transverse slopes are about 14° and 3° respectively.

The best material for beams for structures of types 2, 3 or 4 is reinforced concrete, since it can be so easily molded to any desired detail for support of sheave bearings and can be used equally as well for strut-beams as for beams subjected to bending only. It is also probably

more economical than steel when used in connection with concrete piers.

Turn-sheave Location and Support (By C. R. Forbes).—At a Lake Superior copper mine, a turn sheave was necessary because a suitable place for the hoisting engine could not be obtained in the line of the shaft, at a sufficient distance from the headsheave to permit the rope to wind properly on the drum of 11-ft. face; a minimum distance of 400 ft. was required and at that distance there was situated an office building and boarding house; furthermore the character of the ground at this point was not suitable for constructing a hoisting-engine foundation at a reasonable cost. However, at a point to the east of the shaft, a sufficient

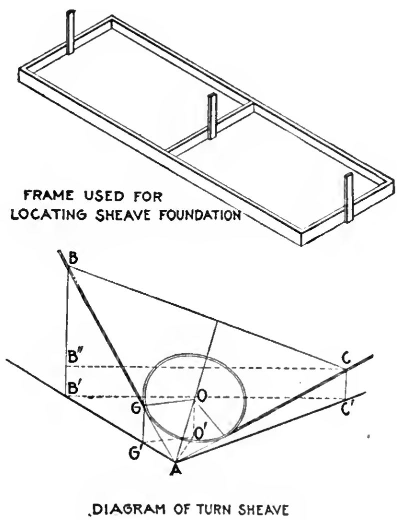


FIG. 239.—TURN SHEAVE LOCATING FRAME AND GUIDING DIAGRAM.

distance away, there was an outcrop of trap rock which made an ideal place for the foundation, but which necessitated the use of a turn sheave shown in the illustrations. The method of supporting this sheave was somewhat out of the ordinary, as the top of the concrete base was made parallel to the plane of the sheave instead of horizontal, as is the more common practice. The timbers supporting the sheave are 14 in. square and are held in place by 1½-in. bolts set in the concrete.

In constructing a foundation of this kind, the most difficult part is in locating and erecting the form for the odd-shaped concrete base. The operation in this case was accomplished as follows. A rectangular frame, as shown in Fig. 239, was constructed of 2 × 4-in. timber, the frame being made the same size as the top of the concrete base. Projecting up from this frame, perpendicular to it, were three upright pieces; the

length of these pieces was the same as the distance from the center of the sheave to top of the concrete, which was determined by the size of the timbers, the length of the hub and shaft of the sheave, etc. It was necessary to use these projecting pieces to line in the frame, as all calculations were made with respect to the plane of the sheave itself and not the top of the concrete.

From a survey, the horizontal angle between the ropes was determined and also the slope of the rope in each direction; and by various calculations, the direction, the length and the slope of the line from the point of intersection of the ropes to the center of the sheave, were obtained. To locate the frame, the transit was set up in the determined line, and the frame lined in by sighting along the tops of the uprights; by measuring to a plumb line suspended from the top of the middle upright, it was centered correctly and placed at the proper height. The slope in both directions had been calculated, and by the use of 2° levels set at the proper angles and placed on the frame, it was possible to set it at the correct slope, where it was braced and held firmly in position. The form was then built around it with a batter of 1:4, and the bolts set in position and held by a template until the work of filling in with concrete was completed. It is possible to locate a sheave of this kind without the use of a transit by simply stretching chalk lines from the point of intersection toward the headsheave and engine drum and lining in the sheave by eye, although this method would not be so accurate as the one described and would take more time on the part of the workmen.

The solution of the turn-sheave problem, although tedious, is comparatively simple and may be accomplished in several ways, probably the easiest of which, and one that involves the use of plane trigonometry only, illustrated in Fig. 239, is as follows: From the survey, the horizontal angle $C' A B'$ between the lines of rope is found to be 106°; the slope of the rope from the point of intersection to the headsheave, 46°; the slope of the rope from the point of intersection to the engine drum, 10°; diameter of the turn sheave is 11 ft. Referring to the diagram, it is readily seen how the true angle between the ropes, the angle CAB, can be determined. This angle is found to be 93° 32′, and the line from the point of intersection A, to the center of the sheave, will bisect this angle, but it must be remembered that the projection of this line on the horizontal plane, the line AO' in figure, will not bisect the angle $C'AB'$, and further calculations must be made to determine the horizontal angle $O'AG'$. It is next necessary to determine the length and slope of the line AO; with the data given in the problem this length is found to be 7.55 ft. and the slope 40° 41′. In order to find the angle $O'AG'$, draw OG, a horizontal line in the plane of the sheave. The length

of this line can be determined, and as it is equal to $O'G'$, it can be used in solving the triangle $O'AG'$, of which three sides will be known; this angle is found to be 65° 49′.

The data necessary for locating the sheave then are: Horizontal angle to be turned at point of intersection from line of rope to headsheave, 65° 49′; horizontal distance from point of intersection to center of turn sheave, 5.72 ft.; vertical distance from point of intersection to center of turn sheave, 4.92 ft.; slope of line from point of intersection to center of turn sheave, 40° 41′; slope of line at right angle to previous line (CB in figure), 22°.

Turn Sheaves at the Lake Mine (By Karl A. May).—Turn sheaves for guiding the hoisting ropes from the headframe sheaves to the hoist drums are in use at the Lake mine, in Michigan. These serve the same purpose as the turn sheaves described by C. R. Forbes, but differ in design and method of installation. The solution of the problem of finding the proper location, elevation and inclination for the sheaves and

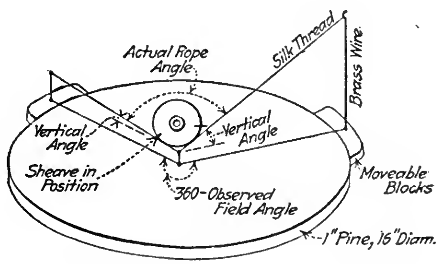


FIG. 240.—MODEL FOR SHEAVE LOCATION.

frames, was essentially the same, but a graphic solution was also made in order to check the computed results, and in addition a model was constructed, the details of which are shown and explained by Fig. 240. The model was put to good use throughout the process, in overcoming some of the difficulties attendant on using the third dimension, and in laying out to scale the existing conditions with a rather high degree of accuracy. The sheaves are set at an angle of about 48° with the horizontal. Separate calculations were made for each sheave, although a difference of only about 30′ in the angle of inclination resulted.

When the calculations were complete, the boss carpenter was given the two center lines of the foundation, a bench mark, and a print showing the location of the anchor bolts, and from these data he built the templates, which consisted of 2-in. planks, in which holes were bored for the anchor bolts. The planks were set to bring the bolts at the proper elevation. The latter were made with the lower end bent to a hook and an old 20-lb. rail was laid in each line of hooks.

The foundation is a level block of concrete with right-angled corners, but set at an angle of 4° 01′ from the line of the rope to the drums, as by this means the sills and the sheave frames are square with each other and with the foundation. The sills and frames are bolted together of 12 × 12-in. Oregon fir and tied with 1½-in. rods. The sheaves are 12 ft. in diameter. The details of the lower bearing, which is important, as it takes the greater part of the load, are shown in Fig. 241. Small guide pulleys are attached to the frames, where the ropes enter and leave the sheaves,

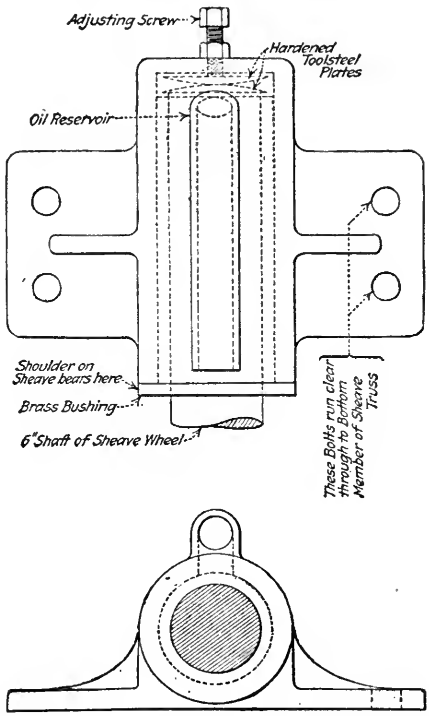


FIG. 241.—LOWER BEARING OF SHEAVE SHAFT.

to take up lash and prevent the ropes from jumping out, due to stretching or to sudden stopping and starting of the hoist.

TRESTLES

Tabulation of Trestle Bent Dimensions (By Clinton Kimball).—The following method of laying out and dimensioning the bents of a tramway trestle, at one of the mines in Wisconsin, was used with consequent saving of time in engineer's field work and in carpenter work. The tramway described is 1250 ft. long and constructed to carry a car

having a gross weight of 2 tons. The bents, as shown in Fig. 242, consist of two 4 × 6-in. inclined posts *A*, with a 6 × 6-in. by 4-ft. cap and 2 × 6-in. diagonal braces *C* and the tie pieces *B*. These bents are spaced 16 ft. between centers and carry 3 × 12-in. stringers; each post rests on a 12 × 12-in. concrete pier extending 2 ft. below the soil surface. Each stringer is spliced over the cap with two 1 × 12-in. by 4-ft. boards and diagonally braced at each end to the posts with 2 × 6-in. by 8-ft. pieces. Bents over 12 ft. high are connected by longitudinal 2 × 6-in. tie pieces placed 6 to 8 ft. below the stringers. Stakes were driven on the center line of the proposed tramway and carefully spaced 16 ft. apart on the horizontal. A nail was driven in each stake to mark accurately the

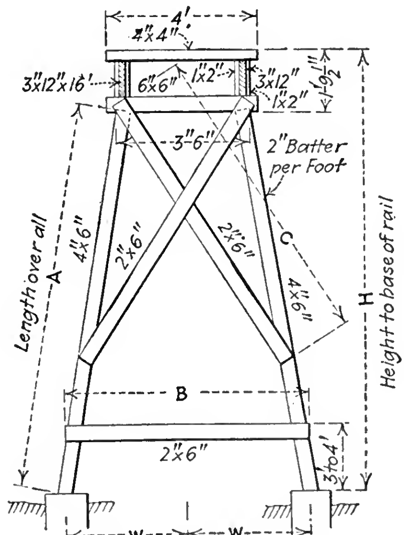


FIG. 242.—BENT, SHOWING CONSTANT AND VARIABLE DIMENSIONS.

center of each trestle bent. A profile of the tops of these stakes indicating tops of piers was plotted from regular level notes. A profile of the grade of the tramway base of the rail was plotted on the same sheet and the height *H* from top of piers to base of rail was thus found. This was later checked by computation. Having the distance *H* for each bent, the dimensions *A*, *B*, *C* and *W* are readily specified, using the accompanying table and interpolating. The other pieces were of constant dimensions. It is desirable to keep the tops of piers at such a height that the pair supporting a bent shall have their tops on the same level and that the one on higher ground shall be nearly flush with the soil surface.

RELATIVE DIMENSIONS OF VARIABLE MEMBERS OF TRESTLE BENT FOR DIFFERENT HEIGHTS

H	A	B	C	W	H	A	B	C	W
Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.	Ft. in.
3	1- 3⅜	Use 4 × 6-in. by 6-ft. sill under posts	3-6	1- 9½	13-4	11- 9⅛	7-0	10-0	3- 6⅛
3-4	1- 7⅜		4-0	1-10⅛	13-8	12- 1⅛	7-0	10-0	3- 6¾
3-8	1-11½		4-0	1-10¾	14	12- 5¼	7-0	10-0	3- 7½
4	2- 3½		4-0	1-11½	14-4	12- 9¼	7-0	10-0	3- 8⅛
4-4	2- 7⅝		4-0	2- 0⅛	14-8	13- 1¼	7-0	10-0	3- 8¾
4-8	2-11¾		4-0	2- 0¾	15	13- 5⅜	7-0	10-0	3- 9½
5	3- 3¾	4- 6	4-0	2- 1½	15-4	13- 9⅜	7-0	10-0	3-10⅛
5-4	3- 7¾	4- 6	4-0	2- 2⅛	15-8	14- 1½	7-0	10-0	3-10¾
5-8	3-11⅞	4- 7	5-0	2- 2¾	16	14- 5½	7-0	10-0	3-11½
6	4- 3⅞	4- 7	5-0	2- 3½	16-4	14- 9⅝	8-0	12-0	4- 0⅛
6-4	4- 7⅞	4- 7	5-0	2- 4⅛	16-8	15- 1⅝	8-0	12-0	4- 0¾
6-8	5- 0	4- 7	5-0	2- 4¾	17	15- 5¾	8-0	12-0	4- 1½
7	5- 4	4- 9	5-6	2- 5½	17-4	15- 9¾	8-0	12-0	4- 2⅛
7-4	5- 8⅛	4- 9	5-6	2- 6⅛	17-8	16- 1¾	8-0	12-0	4- 2¾
7-8	6- 0⅛	4- 9	5-6	2- 6¾	18	16- 5⅞	8-0	12-0	3- 3½
8	6- 4¼	4- 9	5-6	2- 7½	18-4	16- 9⅞	8-0	12-0	4- 4⅛
8-4	6- 8¼	4- 9	5-6	2- 8⅛	18-8	17- 2	8-0	12-0	4- 4¾
8-8	7- 0¼	4- 9	5-6	2- 8¾	19	17- 6	8-0	12-0	4- 5½
9	7- 4⅜	4-11	6-0	2- 9½	19-4	17-10⅛	9-0	14-0	4- 6⅛
9-4	7- 8⅜	4-11	6-0	2-10⅛	19-8	18- 2⅛	9-0	14-0	4- 6¾
9-8	8- 0½	4-11	6-0	2-10¾	20	18- 6¼	9-0	14-0	4- 7½
10	8- 4½	5- 3	6-0	2-11½	20-4	18-10¼	9-0	14-0	4- 8⅛
10-4	8- 8⅝	5- 3	6-0	3- 0⅛	20-8	19- 2⅜	9-0	14-0	4- 8¾
10-8	9- 0⅝	5- 3	6-0	3- 0¾	21	19- 6⅜	9-0	14-0	4- 9½
11	9- 4¾	5- 7	7-0	3- 1½	21-4	19-10⅜	9-0	14-0	4-10⅛
11-4	9- 8¾	5- 7	7-0	3- 2⅛	21-8	20- 2½	8 ft.	12 ft. and a lower diagonal	4-10¾
11-8	10- 0⅞	5- 7	7-0	3- 2¾	22	20- 6½			4-11½
12	10- 4⅞	6- 0	8-0	3- 3½	22-4	20-10⅝			5- 0⅛
12-4	10- 8⅞	6- 0	8-0	3- 4⅛	22-8	21- 2⅝			5- 0¾
12-8	11- 1	6- 0	8-0	3- 4¾	23	21- 0¾			5- 1½
13	11- 5	6- 0	8-0	3- 5½	23-4	21-10¾			5- 2⅛
					23-8	22- 2¾			5- 2¾

It may be seen that the framers were able to proceed with the bents independently of the erectors. The method is recommended even for tramways a few hundred feet in length and especially in cases where an old tramway must be replaced, allowing the minimum of time for the shutdown incident to replacement, since piers can be set and all framework prepared to dimension before interfering with operations.

Typical Coal Trestles.—The surface equipment of many mines includes a trestle for bringing the coal to a point where it will dump into the boiler-house bins and also for providing storage against winter use.

The accompanying illustrations, Fig. 243, show some typical methods of construction employed on the Mesabi range and also give the profile of one trestle. Squared timber for such trestles is almost universal.

Number of bent	Length of inside posts		Length of outside post		Length of sill		Number of bent	Length of inside posts		Length of outside post		Length of sill	
	Ft.	In.	Ft.	In.	Ft.	In.		Ft.	In.	Ft.	In.	Ft.	In.
1	11	9 3/8	12	1 3/8	16	0	12	17	10 3/4	18	4	18	6
2	12	3 3/8	12	7 1/2	16	0	13	17	10	18	3 3/8	18	6
3	16	2 3/4	16	7 3/4	18	0	14	17	9 1/4	18	2 5/8	18	6
4	16	6 3/8	16	11 3/8	18	0	15	17	8 5/8	18	1 7/8	18	6
5	16	10	17	3	18	0	16	17	7 7/8	18	1 1/8	18	6
6	17	5/8	17	5 3/4	18	0	17	17	7 1/8	18	3/8	18	6
7	17	2 3/4	17	8	18	3	18	17	6 3/8	17	11 5/8	18	6
8	17	5	17	10 1/8	18	4	19	17	5 5/8	17	10 7/8	18	6
9	17	7 1/8	18	3/8	18	4	20	17	4 7/8	17	10 1/8	18	6
10	17	9 1/4	18	2 5/8	18	6	21	17	4 1/8	17	9 3/8	18	6
11	17	11 3/8	18	4 3/4	18	6							

In 1 is shown a trestle built on a 16° curve with a 2.5 per cent. grade. The trestle is 350 ft. long and every third panel is braced with 3 × 12-in. diagonals to form towers. The rails are laid directly on stringers which are themselves carried on corbels over the bents and are spliced with steel plates on the sides. The bents are symmetrical and running boards are provided on both sides. In 2 is represented a trestle similar to that of 1. It is unsymmetrical, however, and the running board is on one side only, carried by an extension of the cap. In this trestle, the grade is 4 per cent., except on one portion, which has a 12° curve, where it is 3 per cent. These grades and curves represent the maximum permissible in good design. The bents are spaced 15 ft. 9 in. center to center and every third panel is diagonally braced with 3 × 12-in. planks. The tension rod and filling blocks between the stringers are inserted at every bent on the tangents, while on the curve an additional set is used between the bents.

A bent is shown in 3 in which the rails are carried on crossties, four stringers being used in this case. The trestle is symmetrical, with footboards on both sides; all the posts are inclined. The bents are spaced 16 ft., the long ties 8 ft.; between the long ties are 25-in. ties spaced 2 ft. In this case also every third panel is diagonally braced. In all these three types the safe maximum height is 22 ft. The trestle in 4 is similar to that of 3 in its use of crossties. The bents are spaced 12 ft.; three stringers are used under each rail. In 5 a profile of this trestle is

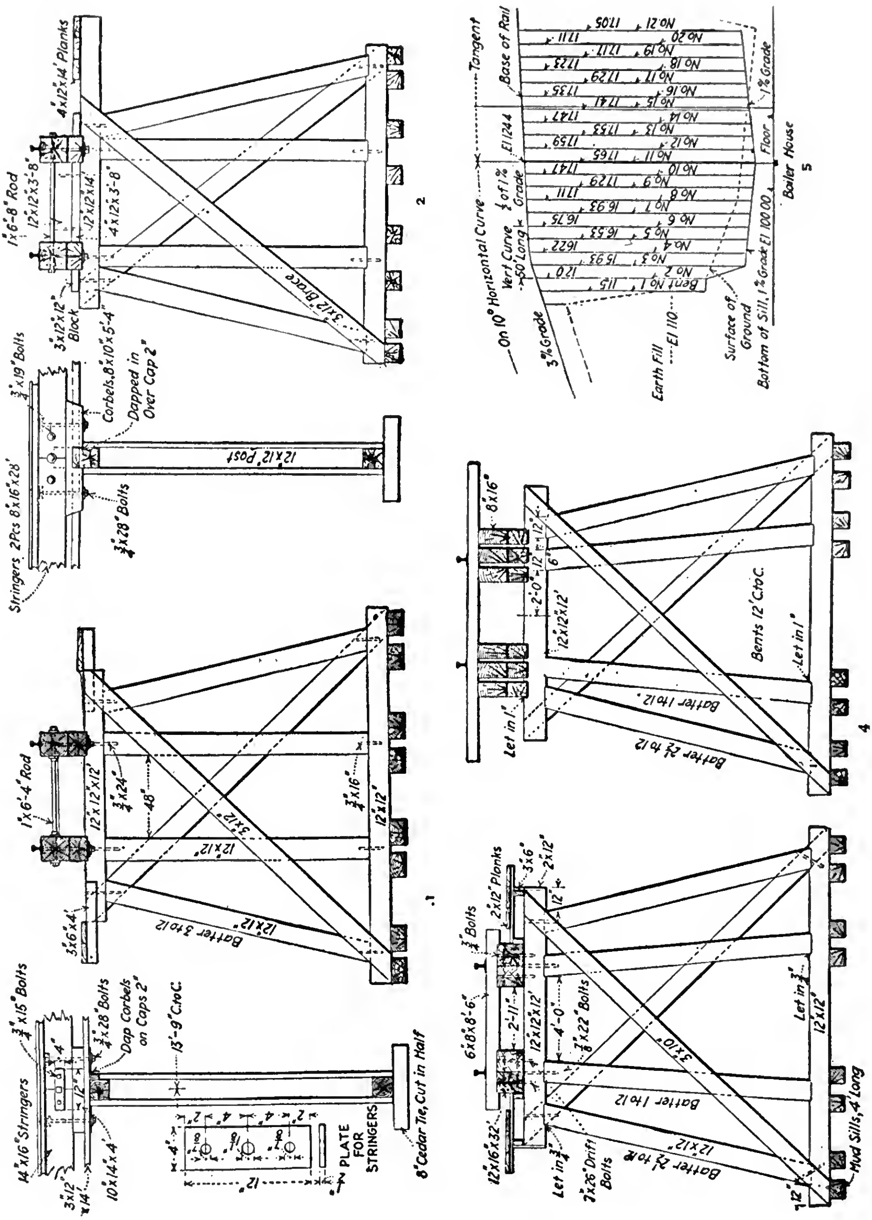


FIG. 243.—SOME VARIATIONS IN DESIGN OF COAL TRESTLES FOR MINES.

shown, indicating the grades and curves and in the table the dimensions for each bent are specified.

Post and Cap Joint for Dumping-trestle.—The material stripped from the Mesabi openpits is disposed of in great dumps, which are built out and up from a single trestle extending the length of the projected dump. This trestle is of light construction, serving only to start operations and is finally buried and left. The method of construction is shown in Fig. 244. The first material stripped is dumped in

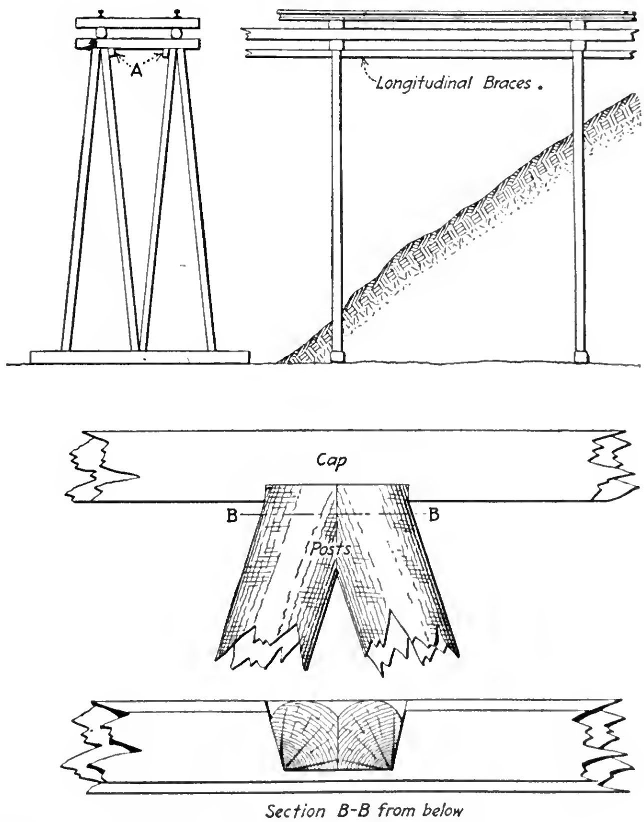


FIG. 244.—ELEVATIONS OF TRESTLE END AND DETAIL OF NEW JOINT.

and around the trestle so as to make it a fill. Dumping begins at the end of the trestle near the pit. The train approaches the dump with the engine behind; when the first car is at the end of the filled part of the trestle, it is dumped, pushed ahead on the trestle and the next car dumped. In this way the light trestle has to support empty cars only. There is always an inclined face or toe to the fill which exerts a considerable pressure on the posts of the bents which it surrounds, as shown in the

illustration. The tendency is to push the posts out from under the caps. To prevent this, longitudinal braces are usually set between the posts under the stringers. At the Leonard mine in a recent trestle, these braces were omitted and the posts framed into the caps for an inch or two in a sort of box, the joint A, indicated in the front elevation of the bent, being made as shown in detail in the lower part of the illustration. This evidently prevents the posts from being pushed through, and in practice has proved satisfactory. It saves the entire cost of the braces and also the labor of setting them, while the labor of framing in this manner is little more than for the former method.

Wood Trestle for Motor Tramming.—From the typical iron-range shaft several trestles usually lead off, one or two for the ore stockpile,

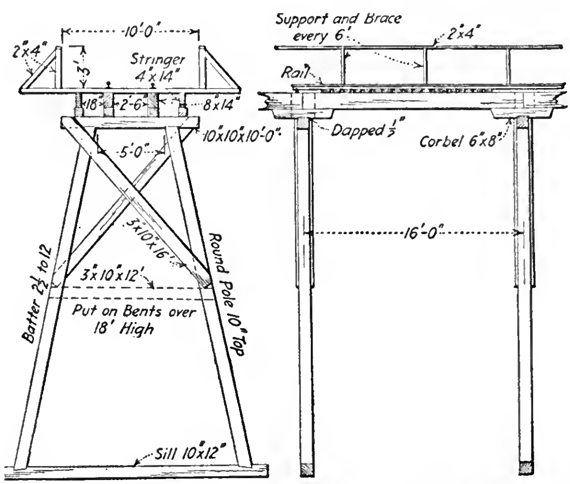


FIG. 245.—CONSTRUCTION OF A TYPICAL TRAMMING TRESTLE.

one for the waste rock and a short one for a tail track, when motor tramming is employed. The bents of these trestles are sometimes steel, more frequently wood, and in the latter case are often temporary. The accompanying illustration, Fig. 245, represents a rather typical bent of light construction. It is designed for a load of 10 tons, that is, a loaded car and an electric locomotive. The material specified is hemlock, except for the posts which are cedar, spruce, or tamarack round poles. Such a trestle will range in height from 15 to 50 ft., although for the higher structures, three or four posts are used in the bent. The stringers, instead of being sawed timber, are frequently hewed poles, which may be lapped and the corbels eliminated. The full sill shown here is often omitted on a temporary bent and 1 × 6-ft. bearing blocks used under the various posts.

Permanent Stockpile Trestle of Wood (By Oscar Gustafson).—The Colby Iron Mining Co., operating the Colby and Ironton mines on the Gogebic range, has in use a wooden stockpile trestle, which is designed to be permanent and is giving great satisfaction. Besides not having to be taken down each year, it differs from the trestle in common use chiefly in that the bents have but one leg instead of two. Loading tracks are laid on each side of the stockpile, and the one line of legs spaced 32 ft. apart offers but little difficulty to steam-shovel loading. It is an advantage to both the railroad and mine to have a loading track on each side of the stockpile, for after finishing the first cut the end of the loading track is thrown over and the shovel runs back on this track and starts a

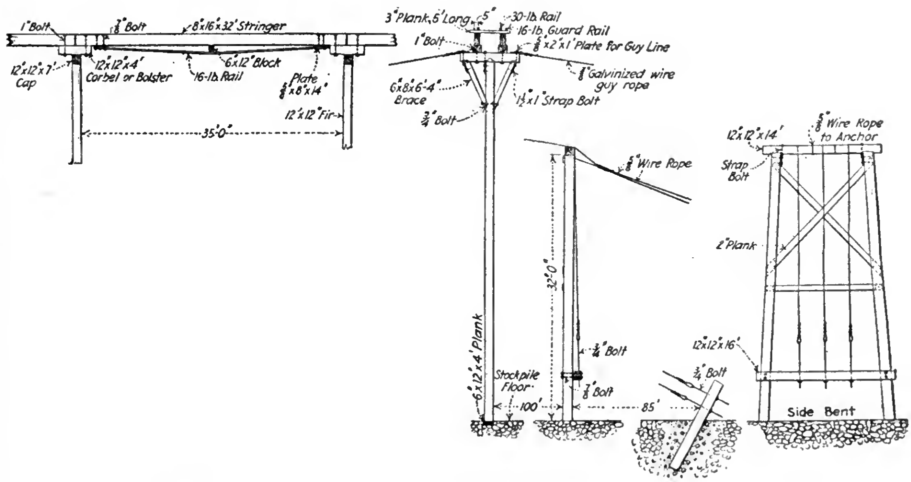


FIG. 246.—ELEVATIONS OF MAIN BENT, SIDE BENT AND ANCHORAGE.

cut on the other side of the pile without waiting for the first track to be placed again and connected. This saves considerable time to the shovel and crew. While the shovel is working on the second cut, the railroad section crew has plenty of time to place the first track and is not compelled to call in an extra gang. The trestle is found to be cheaper than the old trestle, remains in alignment better while loading is going on, and is more sightly.

The design follows broadly that of the Negaunee permanent steel trestle. Each bent is a single leg of 12 × 12-in. fir, 38 ft. long, on which a 12 × 12-in. fir cap, 7 ft. long, is mounted and braced by two 6 × 8-in. by 6-ft. fir braces, mortised and bolted to both leg and cap. To each cap are bolted two 12 × 12-in. by 4-ft. fir corbels or bolsters, to which again are bolted the 8 × 16-in. by 32-ft. fir stringers, as shown in Fig. 246. The stringers are trussed with 16-lb. rails; to each end of these a 5/8-in.

plate is riveted and then bolted to the stringer. The truss rods are blocked in the center with a 6 × 12-in. wood piece. To the stringers are spiked 3-in. planks 5 ft. long, and the 30-lb. rails are laid on the planks at 30-in. gage. Outside of the 30-lb. rail, a 16-lb. guard rail is spiked.

To each end of the cap is bolted a plate with an eye in the end, for attaching the guys. These guys are ⅜-in. galvanized-wire strands; they extend out to side bents erected at 100 ft. from the trestle, the guys from three center bents being attached to each side bent. The guys pass over the cap and down to eye-bolts, passing through a 12 × 12-in. by 16-ft. timber near the ground. The side bents are 32 ft. high, built of round timber and well braced. They are themselves guyed by two ⅝-in. wire-rope guys to a "deadman," concreted in the ground.

Raising Trestle without Ginpole (By R. B. Wallace).—The apparatus shown in Fig. 247 was devised at the Republic iron mine in Michigan

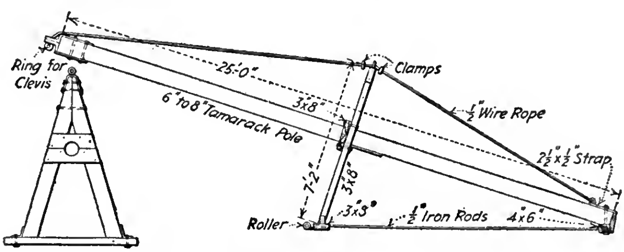


FIG. 247.—MOVABLE TRUSSED POLE FOR RAISING TRESTLE BENTS.

to facilitate the erection of a stockpile trestle, by eliminating the operations of moving a ginpole, changing guys, etc., which consume much time. It consists of a sound tamarack pole 6 or 8 in. in diameter, supported by an A-frame at the center and trussed by a ½-in. wire rope attached to both ends and passing over the top of the frame. The base of the frame and the crosspiece at the lower end of the pole are connected by two ½-in. rods. A roller made of wood with both ends incased in pipe is attached to the base of the frame and in front of it, so that by raising up the lower end of the pole, the whole apparatus can be easily rolled out to the end of a trestle. With the lower end of the pole lashed to the rails or to a stringer, it is ready for use in hauling up a bent. After the bent is hauled up, it is held by guy lines while the stringers are raised and fastened in place. Then the frame is rolled out to the new end of the trestle, and is made fast, ready for the next bent. An electric winch is used for power.

Erecting Trestle Bents with Cableway (By A. Livingstone Oke).—The diagram, Fig. 248, shows a convenient method adopted for erecting quickly a long series of trestles to carry an inclined gravity tramway down a steep hillside. The carpenters built the trestles on the site,

starting with the last on the lower end of the section and stacking them in ascending order. Meantime, a wire-rope cableway was erected by means of two shears, so that the lowest point of the rope was above the finished tramway level. A winch was set at the upper end, as shown. The winch-line operated over a carrier sheave running on the fixed rope; it had a weighted hook at its end and a stop placed a few feet above it. The hook was used to take hold of the bent tops and the rope pulled by the winch raised the bent. The stop, catching on the sheave, brought

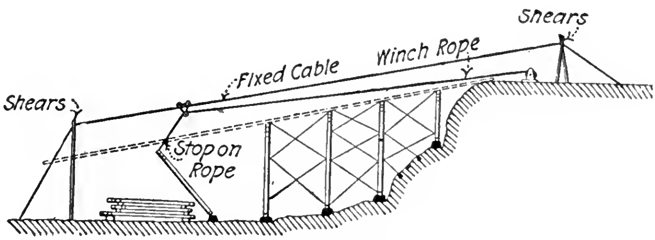


FIG. 248.—PROFILE OF TRESTLE AND CABLEWAY.

the bent clear back to the required position, once it was swinging clear of the ground. Each bent was temporarily fastened in a vertical position by extending running planks from the top of the bent last erected, until the permanent bracing was put in.

GINPOLES

A Built-up Ginpole (By A. Livingstone Oke).—It frequently happens, where timber is scarce, that a ginpole or other long timber is required

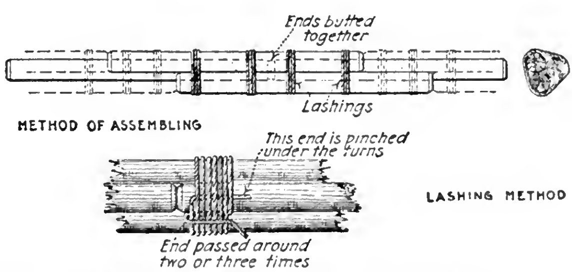


FIG. 249.—THREE ROUND TIMBERS LASHED TOGETHER FOR A GINPOLE.

larger than any single stick on hand. A method of building one from short lengths of either round or square timber is exhibited in Fig. 249. The poles are laid up in threes, the ends overlapping, as shown, with a lashing on both sides of each joint; the number of turns of rope used depends on whether the pole has to sustain merely a thrust, as in a ginpole, or is to be employed for work in which it will be subjected to

bending stresses. In the latter case, both the strength of the ropes, the number of turns and the tightness with which they are put on, must all be increased. If the poles available are all of the same length, but of different diameters, some care must be used in matching the sizes at different points in the length of the built-up pole, keeping the stouter ones together and gradually tapering to the smaller end. Poles may also be built with more than three members, but this number is the more satisfactory, when possible, as it makes a three-point support inside and prevents transverse movement. The proper method of lashing is also shown; several turns of small rope are to be preferred to fewer turns of a larger rope. One end of the rope is pinched under the turns and finally the free end taken around these by passing through several times, as shown, and the whole drawn up tight. To tighten the lashing further, short lengths of smaller rope may be passed around the turns on the other two sides as well and drawn as taut as possible.

Ginpole of 10-in. Pipe.—A ginpole used by the Oliver company in the Chisholm district on the Mesabi range in constructing its stockpile trestles is herewith illustrated. It is built of 10-in. pipe reinforced against possible buckling by pieces of round fir about 3 ft. long inserted three to a length of pipe and bolted to keep them from slipping. The stockpile trestles run from 40 to 50 ft. in height usually and the ginpole will reach perhaps 20 ft. above the floor of the trestle. Details of the pole are shown in Fig. 250. About 5 ft. from the top a horizontal arm is rigidly attached; it is made of a stout T-rail and is held to the pole by an iron strap and also by an inclined stay-rod of round iron, yoked at the top to embrace the top of the pole, to which it is bolted, and looped under the head of the rail at its lower end. The web of the rail is cut away to permit this looping and the head and base held together by a bolt. Near each end of the rail a clip is bolted, the upper part being a clevis to embrace the web and base of the rail and the lower part pierced to allow blocks to be hung. One double and one single block are used for hoisting.

About 20 ft. down from the top the pole is held against the trestle by a horizontal timber, 6 × 10 in. in section and about 10 ft. long, tapered somewhat at the end next the pole and framed to fit the pole; the bearing is completed at this point by a ½ × 6-in. strap bolted to the timber and passing around the pole, thus permitting rotation. The timber is fastened to the decking of the trestle. The lower end of the ginpole is closed with a plate having a hole in the center to fit a foot-piece bearing. This bearing piece is a casting with a square base containing four bolt holes and a vertical cylindrical portion. About 4 ft. above the bearing two opposite holes in the pipe permit the insertion of a bar with which the pole is turned on the bearing described. A rough supporting platform is built of heavy planks and hewn timber.

This ginpole is not used for erecting the trestle bents, which are framed on the ground and hoisted with a wooden pole, but a great deal of material goes into the trestle, such as flooring, rails, railings, trolley-wire supports, etc., and there is no way of getting this up from the ground

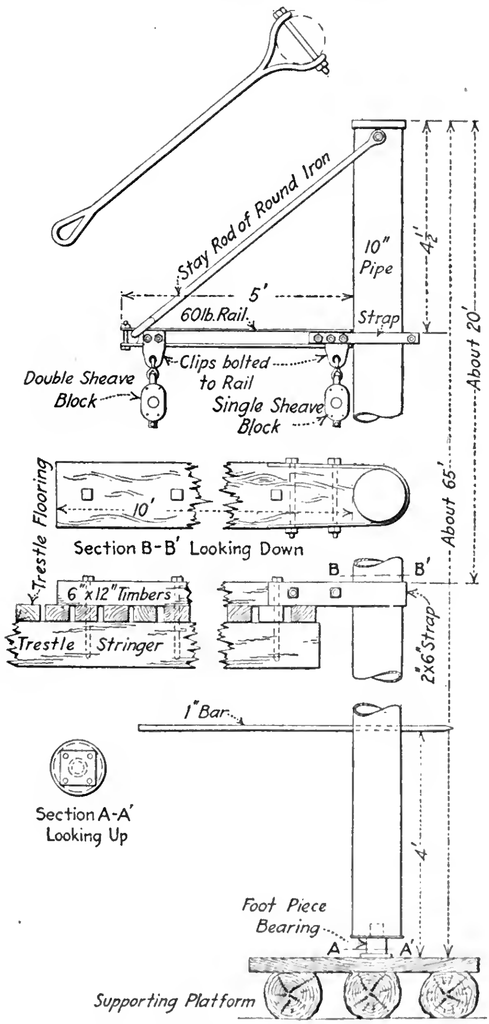


FIG. 250.—DETAILS OF GINPOLE RIGGED FOR HOISTING.

except by the use of some special device, since the shafts are not equipped with cages and the skips are inconvenient as well as being usually otherwise engaged.

Handling Stack with a Ginpole (By A. Livingstone Oke).—The proper method of placing stacks or columns on their foundations, using

a single ginpole is shown in Fig. 251. While not new, it is probably not known to all. There should be two guy ropes at the back, secured firmly, so that the pole has a good cant toward the pull of the tackle. The lashing must be placed on the stack as far above the center of gravity as the available height of the ginpole allows, the distance *a* always being kept greater than *b*, after making due allowance for the length taken up by the tackle itself. The power may be applied from a windlass or by direct pull, depending on the weight to be handled. The more nearly vertical the pull the better, a snatch-block being necessary if the windlass is not directly below the lifting tackle.

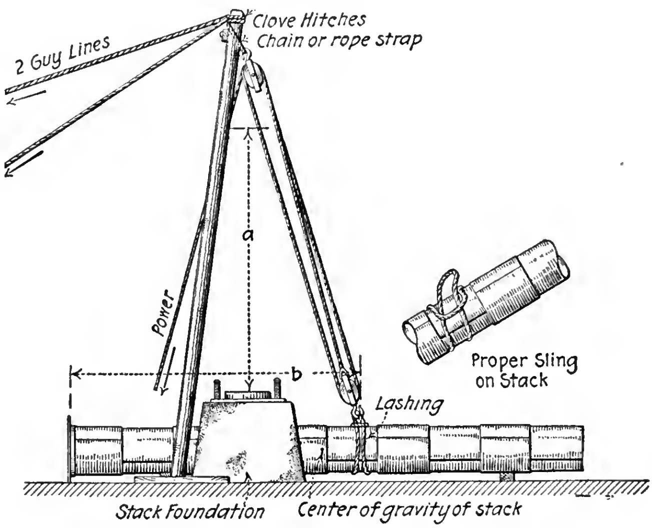


FIG. 251.—RIGGING OF GINPOLE AND TACKLE FOR SETTING STEEL STACK.

VARIOUS DEVICES

Substructure of Wooden Water Tank.—A necessary adjunct to every Iron-Range boiler planti s the water tank. The flat topography usually eliminates the possibility of setting the tank on high ground to gain head, and for that reason it is usually put near the boiler house. The latest tanks are of the elliptical-bottom steel type, raised to considerable heights on steel structures, but wooden tanks supported on wooden substructures in the manner of railway tanks are still common. Such a substructure of standard type is illustrated in Fig. 252. It is built of sawed timber set on concrete piers and usually painted. The tank supported in this case is 12 × 12 ft.

Surface Steam-line Supports.—In the Norrie group of mines at Ironwood, Mich., steam is conveyed some distance, from the boiler house to

the hoist at one of the shafts. The outdoor steam pipe is carried on structural-steel bents in order to maintain the desired grade over the variable ground. The bents are spaced about 15 ft.; at every 200 ft. approxi-

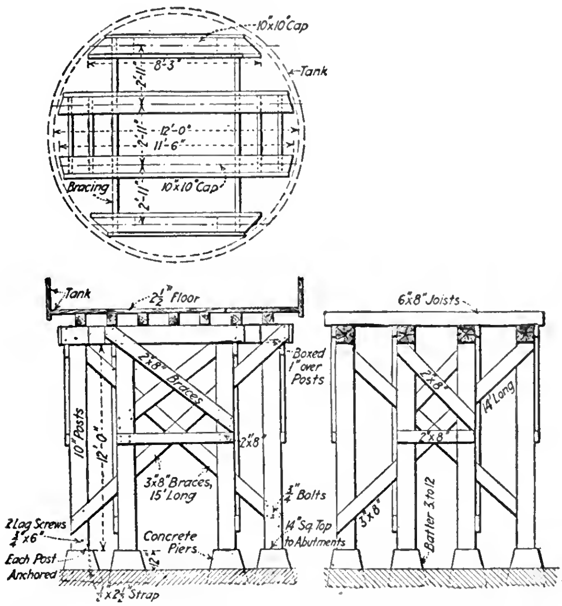


FIG. 252.—POSTS, STRINGERS AND BRACING TO CARRY WOODEN TANK.

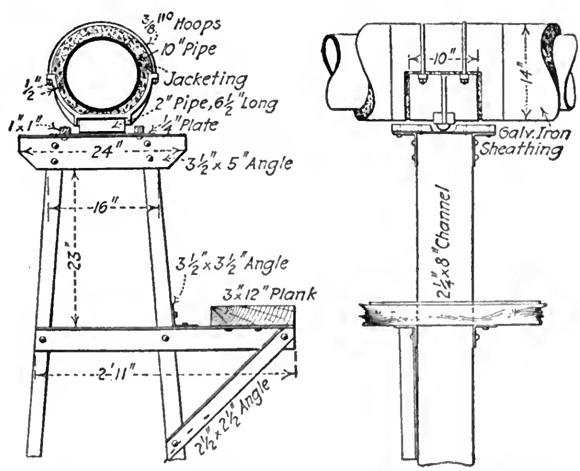


FIG. 253.—TOP OF A BENT CARRYING A STEAM PIPE.

mately a slip joint is inserted to take care of expansion. This is carried on a more elaborate structure. The bents are set on concrete foundations to which they are held by plates bent at an angle, riveted to the legs

and bolted to the concrete. A 3 × 12-in. plank, supported on angles along one side, serves as a walk when it is necessary to inspect the pipe or pack the slip joints. The cross-braces are spaced 3½ ft. center to center and as many are used as the height of the bent requires. The manner of holding the pipe on the top of the bent so as to permit longitudinal motion, as shown in Fig. 253, is of interest. The pipe is jacketed with insulating material and sheathed in galvanized iron, wired on. A cast saddle at each bent is bolted to the bottom of the pipe with two ⅜-in. bolts over the top. The bottom of this casting is a socket which fits over a short section of 2-in. pipe. This pipe acts as a roller; it is free to roll longitudinally on a plate which forms the top of the bent,

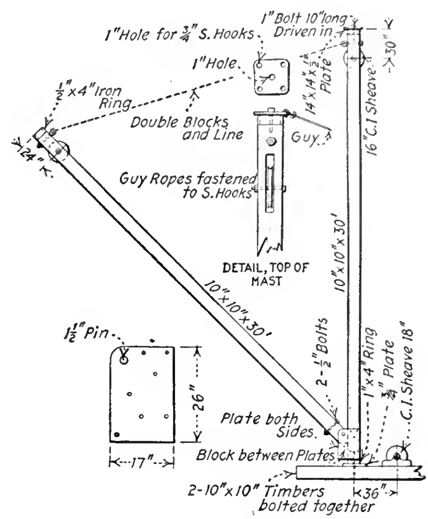


FIG. 254.—DETAILS AND ASSEMBLY OF DERRICK.

but is held against lateral motion by 1 × 1-in. pieces riveted to the plate. The details of construction of the top of the bent are evident from the drawing.

Simple Guyed Derrick (By H. L. Botsford).—The derrick shown in Fig. 254 is one readily constructed by a mine carpenter and blacksmith. It is useful for handling buckets at the beginning of shaft sinking operations, as well as for many other purposes about a mine.

Dipping Tank for Mine Timber (By L. O. Kellogg).—At the Bennett mine, on the Mesabi, Carbolineum is rather extensively used for treating timber for both underground and surface construction. The material for such surface structures as the coal trestle and the casing on the vertical water-tank pipe and for the shaft sets and station timbering underground is dipped in the preservative before it is put in place. The

tank used for the purpose, shown in Fig. 255, is simple enough, but has proved convenient and is so readily constructed as to merit description. It is built of 2-in. lumber throughout, except that the two end sills are 4 × 4 in. Its length is such as to accommodate the wall plates of the shaft sets, the longest pieces which require treatment. The boards in the box itself are tongue-and-grooved and are bolted together to get tight joints. A longitudinal, vertical partition divides the tank into two compartments, one for dipping, and other for draining, the former

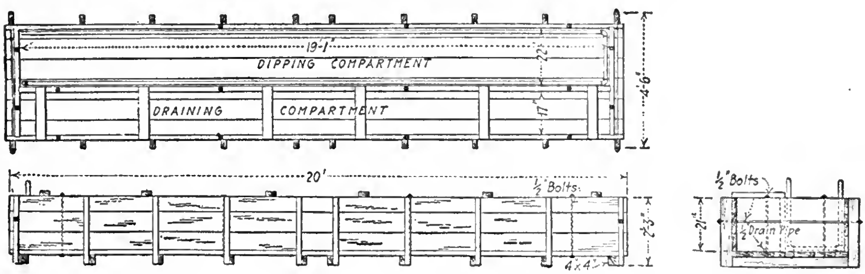


FIG. 255.—WOODEN TANK FOR TREATING TIMBER WITH PRESERVATIVE.

somewhat the larger. The box is bound with sets of 2 × 4-in. material, consisting of a sill and two posts each, spaced about 2 ft. apart. The dipping compartment is lined bottom and sides to a height of about 18 in. with galvanized iron nailed to the wood and soldered at the joints. A steam coil on the bottom along the sides and one end keeps the Carbolineum slightly warm, to obtain a more rapid penetration. The timber is allowed to soak for a few minutes and then lifted out to drain on several 2 × 4-in. crosspieces above the drain compartment.

VIII

HOISTING, LOWERING, TRANSPORTING

Hoisting Devices—Lowering Devices—Signaling—Windlass and Whim —Rollers and Sheaves—Rope—Transporting—Accessories

HOISTING DEVICES

Cableway-and-carrier Bucket Hoist (By R. B. Wallace).—An unusual hoisting arrangement is in use for some small-scale work at the property of the Republic Iron Co., Republic, Mich. A cable is stretched across an old openpit from the tower to a point over the shaft opening which lies on the other side and at the bottom of the pit. A cableway

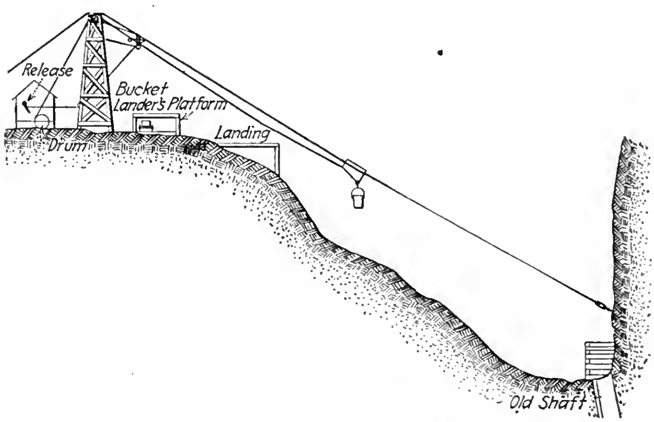


FIG. 256.—LAYOUT OF CABLEWAY, HEADFRAME, HOIST AND SHAFT.

carriage runs across the pit on this cable and carries the bucket; at the lower end it automatically releases the bucket and drops it into the shaft. The general relations are shown in Fig. 256. This shaft descends at about 85°, and enters some old workings where floors and pillars are being cleaned up. The hoisting is done by electricity and the bucket is landed by the engineer. At the completion of a hoist, the carriage is automatically held stationary while the bucket is allowed to descend to the lander's platform. When the bucket has been hoisted up to the carriage again, the moving of a lever in the engine house releases the carriage. Two buckets are used, one being filled while the other is

hoisted. The diameter of the track rope is $1\frac{1}{8}$ in., and of the hoist rope, $\frac{5}{8}$ in. The capacity of the bucket is 0.6 ton. In an awkward situation, such as this, where the quantity of ore does not warrant an elaborate hoisting plant, this scheme works satisfactorily.

Otis Elevator Mine Hoist (By L. E. Ives).—The Naumkeag Copper Co. about a mile west of Houghton, Mich., started exploratory work with an adit, the mouth of which is about on a level with the wagon road and about 75 ft. from it. Because of the lack of elevation and the proximity of the road, dump room at the mouth of the adit was difficult to obtain. The difficulty was overcome in a rather novel and effectual manner.

An artificial single-compartment shaft was constructed by the Otis Elevator Co., over the adit at its mouth. The framework of the shaft was built up of flat, rough boards, approximately $7\frac{1}{2}$ in. wide and $1\frac{3}{4}$ in. thick. Five of these, laid flat sides together and standing on end constituted each corner post, these being about $8\frac{1}{4}$ in. thick. The sides were divided in panels by horizontal members of similar construction, each panel braced diagonally. This framework was inclosed by matched, planed boards $8 \times \frac{3}{4}$ in., the whole making up the boxed-in shaft, 9×10 ft. 7 in. outside dimensions. The lift from the floor of the tunnel to the level of the dump was 30 ft.

The elevator car was nothing more than an ordinary freight-elevator platform with a 2-ft. gage track running across it and two sides boarded to a height of 5 ft., the car itself being 8 ft. long. The hoisting was done by a 10-hp., alternating-current, 230-volt, 2-phase, 60-cycle motor, placed in a separate compartment over the sheaves at the top of the shaft. Two $\frac{3}{4}$-in. steel hoisting cables were used and the hoisting speed was 40 ft. per minute. The tram car held 30 cu. ft., and the hoist was capable of lifting 4000 lb., in addition to the elevator car. The entire installation was operated by a hand cable, the stopping being automatic at both top and bottom.

Power for operating was obtained from the service wires of the Houghton County Electric Light Co., not over 100 ft. away. When hoisting about 600 tons of rock per month the power consumption was about 58 kw.-hr. per month, at a total cost of \$1.80. This figures out about 0.3 cts. per ton hoisted. No hoistmen were required. Two trammers pushed the car out to the shaft and to the platform, pulled the cable, went up with the load, dumped it and returned to the muck pile again. Power was not wasted when no hoisting was going on, and three safety stops guarded against the possibility of overwinding.

Chain-driven Convertible Hoist (By E. E. Carter).—A steam-driven, double-drum, geared hoist at the Gold Hill mine, Quartzburg, Idaho, was changed into an electric hoist by disconnecting the driving and valve

rods and substituting a gear on the crank disk. With this combination the hoisting speed was 450 ft. per minute. The motor ran at 850 r.p.m., and both motor and hoist were operated considerably below rated capacity. Steel and rawhide pinions were tried and found unsatisfactory. Both gear and pinion gave constant trouble from excessive wear, the teeth in the case of the steel gears becoming crystallized and falling out. Furthermore, the vibration was carried to the motor and thus resulted in the motor connections shaking loose, making it necessary to keep an extra one on hand for frequent changes.

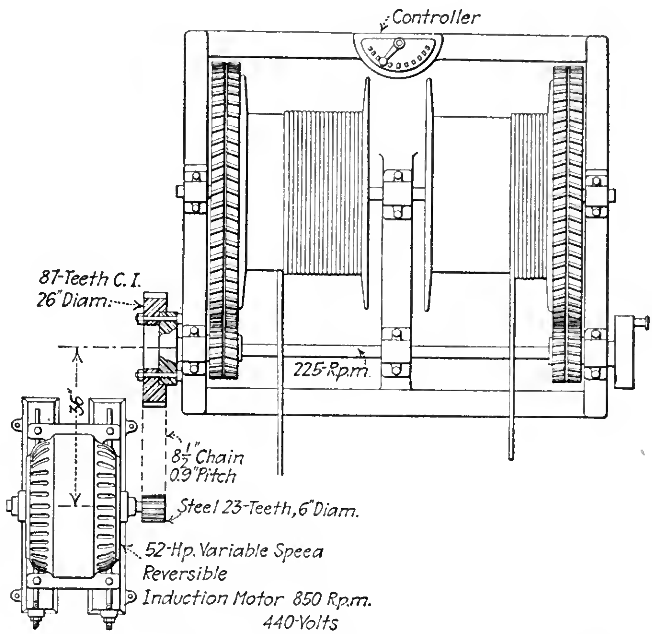


FIG. 257.—STEAM HOIST ARRANGED FOR ELECTRIC DRIVE.

This arrangement was a constant source of expense, as well as a loss of time; therefore, the following arrangement was tried with success: An $8\frac{1}{2}$-in. silent-chain drive was substituted for the gears, set at 36-in. centers, and arranged as shown in Fig. 257. After this drive had been in operation for sufficient time to prove its efficiency, no sign of wear was apparent and no time had been lost because of mishap to the hoist after its installation. The hoist picked up its load much easier than it did with the gears or even better than it did with steam. There was no trouble with the "flop" of the chain, when reversing, as was anticipated.

The cost of the chain installation was practically the same as for the original gear drive. Experience elsewhere has shown that the wear on

these chains is confined almost entirely to the pins. As the chain lengthens because of wear and the pitch becomes greater, the chain will ride higher on the gears and thus give warning a long time before it is necessary to put in new pins. The cost of new pins is nominal and they may be put in easily. This chain was expected to last between three and four years before repairing with new pins.

The 52-hp., variable-speed, 440-volt, alternating-current motor was set on a concrete foundation with a subbase and suitable slide to allow adjustment of the chain. The chain ran comparatively loose. A hoisting speed of more than 600 ft. per minute was the rule instead of 450 ft. as formerly. The chain was removed at intervals and cleaned thoroughly with gasoline, then allowed to dry, and lubricated with Albany, Keystone or a suitable cup grease, previously warmed so as to work in around the pins; any excess was wiped off. It was lubricated at intervals as required with the same lubricant. Mica axle-grease or graphite should not be used. The possibility of disconnecting the gear from the crank disk and replacing the driving rods, etc., allows a change from electric to steam or air power within a short time, which is desirable when power troubles are frequent.

Balancing Dummy in Inclined Shaft (By L. Hall Goodwin).—In the inclined shafts of the Lake Superior district it is the custom to hoist in balance; the practice, however, is different from that used generally throughout the West in that the drums are not designed to clutch in and out of balance but both ropes are wound on the same drum so that the position of one skip in the shaft relative to the other is always fixed. Mines in the development stage usually sink and timber the two-hoisting-compartment shaft in final form, except that rail stringers are put in one road only and hoisting is carried on unbalanced until the shaft has attained a considerable production, when rails are put in the other road and the second skip put in service. At the Houghton Copper mine, a small development property on the Superior lode just north of the Superior mine, a novel method of securing balanced hoisting from the start was employed.

Track stringers were laid in the south road only, but in the north road rails of usual weight were laid on the 12 × 12-in. sleepers to carry a dummy. These rails were laid in the center of the roadway with a 2-ft. gage, so that when permanent stringers were put in they would be laid outside of the present rails, and the latter would be simply transferred to them as the work progressed. The dummy was made of riveted sheet iron and filled with iron scrap, being 2 × 2 ft. in section and 6 ft. long. A point in its construction was a pivoted rear axle, which swung freely on a pinion projecting from the lower end of the dummy. This allowed the four wheels to accommodate themselves to irregularities in the

narrow track, which in this improvised arrangement were probably pronounced, and minimized the possibility of the dummy's jumping the rails.

The main reason for the adoption of this device was the need of cutting down the heavy power demand when starting an unbalanced skip. The contract with the power company imposed a penalty for peak load at all times and an additional penalty for load on the company's lighting peak. The penalty would be nearly four times as great with an unbalanced skip as with the arrangement described, and a larger hoist motor would also be required.

Fleeting Device for Hoist with Conical Drums (By F. H. Armstrong).—On the Brier Hill hoist of the Penn Iron Mining Co., Vulcan,

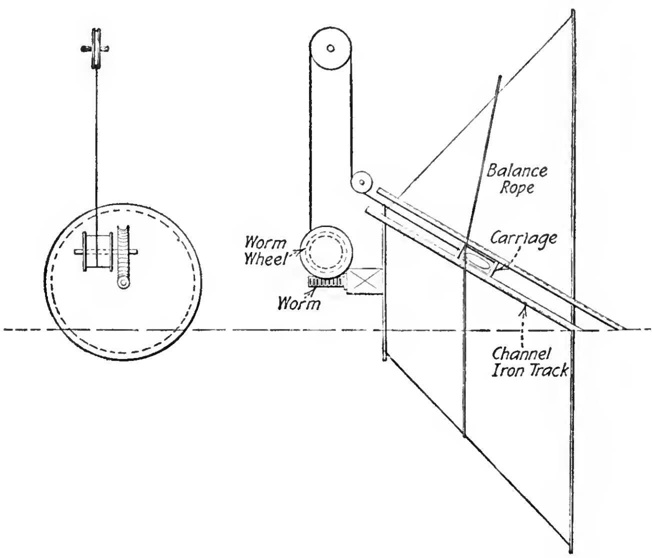


FIG. 258.—DIAGRAM OF FLEETING DEVICE.

Mich., are two 12-ft. cylindrical drums, one for the skip and the other for the cage. There are also two conical drums, from 4½ to 17 ft. in diameter, which are used for counterbalancing the dead loads. The hoist is located so near to the headframe that the sheave to which the counterbalance rope leads is in line with one end of the conical drum, thus pulling heavily sidewise on the deep grooves. To correct this a channel-iron track was set up in the hoist house, in which runs a carriage carrying a sheave. The carriage and sheave are pulled up the track by a rope leading from the carriage to a drum that is driven by a worm and wormwheel from the main shaft of the conical drum. The speed of this small drum is such that it pulls the carriage across the face of the

conical drum the distance between two adjacent grooves for every complete revolution of the conical drum. Fig. 258 shows the carriage, sheave, track and rope leading to the small drum.

Automatic Skip Recorder.—The Franklin Mining Co., in the Lake Superior copper district, is using on its hoist at the Franklin Jr. mine an automatic recording device which provides a permanent record, on paper, of every movement of the skip during a 12-hr. period. The mechanism is comparatively simple. Two drums are mounted on vertical spindles, about 12 in. apart, and these drums contain a paper chart. The paper travels from right to left, the left-hand drum being driven by clockwork. The chart is divided longitudinally according to hours and minutes, from 6 a.m. to 7 p.m., or *vice versa*. A vertical arm, bearing an ink pen, containing red ink, registers both horizontal and vertical lines on this chart, and receives its motion through a chain sprocket and gears from the countershaft of the hoist. There is also a lower arm on the machine which can be connected to the electric-signal system so as to operate a pencil vertically across the lower line of figures on the chart, and thus show the time at which any signals are rung.

Before the device was installed, there was no way of checking conflicting statements of the hoisting engineer and the men underground, in disputes arising as to the position of the skip at any time. In cases where hoisting was held up, the underground dumpers would be likely to say that they could not get the skip, which was lying idle at some level above or below them. The engineer, on the other hand, would claim that the signals were not rung and that there was nothing wrong with the skip. With this recorder in operation, it is impossible for any such dispute to arise. The chart, as illustrated in Fig. 259, shows exactly where the skip was at any hour or minute of the day and how long it took to make any trip; the speed of hoisting or lowering can be computed from the figures, since the vertical divisions on the chart show the positions of the various levels in the mine. The only attention the recorder needs is the filling of the pen with ink every day and the occasional oiling of the parts. The engineers like the device because it places the responsibility where it belongs. The charts are removed daily and filed in the office of the superintendent, where they are kept for several months. The recorder is the invention of J. M. & O. R. Johnson, Ishpeming, Mich.

LOWERING DEVICES

Go-devil Incline Plane.—At the North Star mine, Grass Valley, Calif., the narrow quartz vein, dipping only 23°, is developed by main levels at 300-ft. intervals on the dip. The ore is conveyed to small

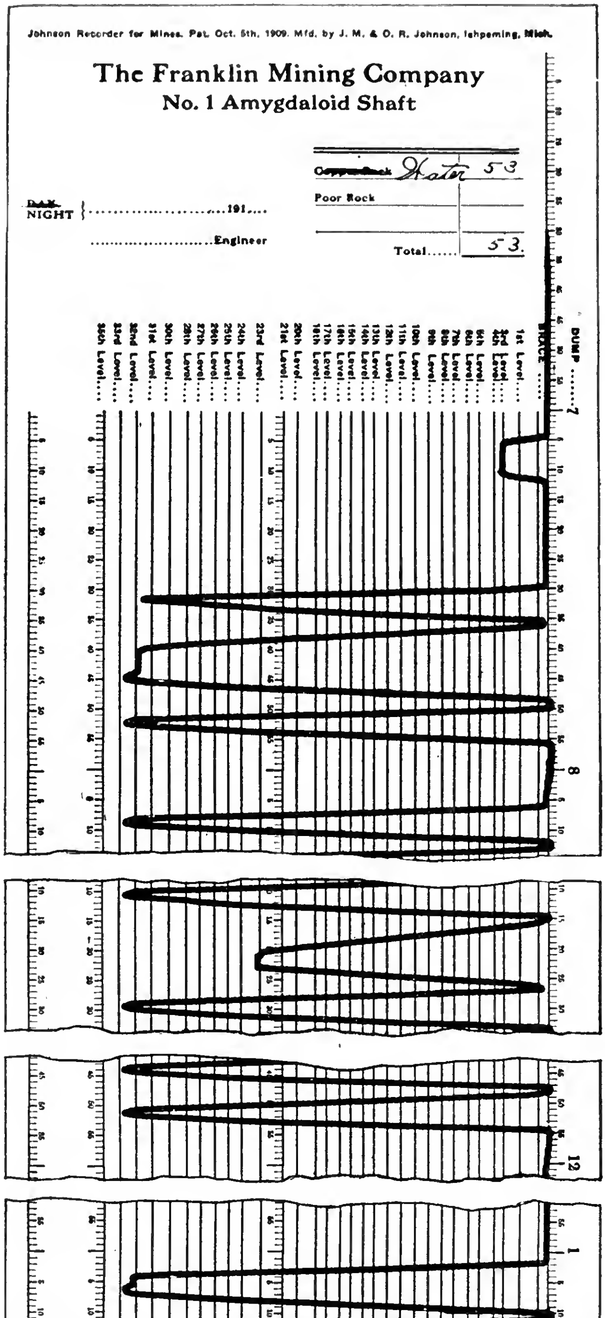


FIG. 259.—SAMPLE OF GRAPHIC RECORD, BREAKS REPRESENTING CURVES OF UNINTERRUPTED HOISTING.

bins along these levels by means of gravity planes through the stopes. As the stope is carried up, the incline track is extended, the headsheave block being kept at the top of the stope, hung from a heavy stull. Tracks branch off horizontally in both directions from the head of the incline, a turnsheet being set between the top of the incline and the head-block post. The cars that are filled along the horizontal tracks are brought to the turnsheet, attached to the rope and lowered in balance against the car last sent down and still attached, empty, to the other end of the rope at the bottom of the incline. The whole system is called a "go-devil."

The head-block is probably the most interesting feature. It is illustrated in Fig. 260. The rope passes up on the outside of the outside

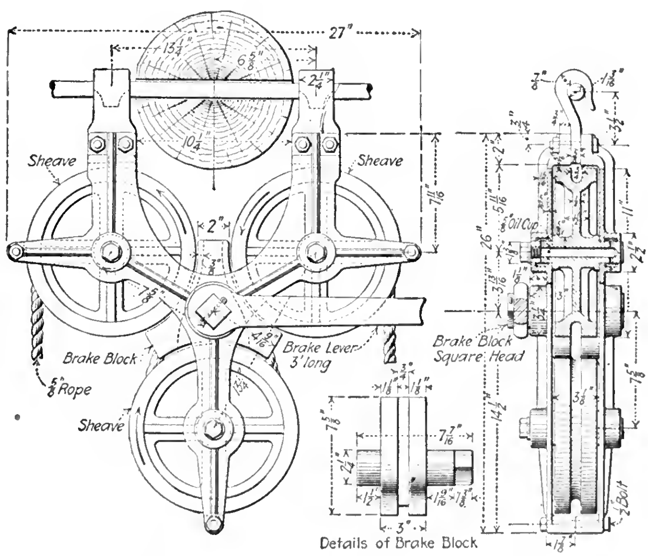


FIG. 260.—TRIPLE-SHEAVE HEAD BLOCK FOR GRAVITY PLANE.

blocks, down the inside through grooves in the brake block, and around the bottom of the lower sheave. The brake block is triangular, pivots about its center in between the three sheaves and is operated by a handle attached to the square head shown. It is brought to bear on all three sheaves and thus gives a strong braking effect. It can be locked in position by extending the handle with a piece of pipe and connecting the end of this to a lever with a ratchet lock set on a stout post. The two halves of the triangular spider-frame, the three sheaves and the brake block make six castings entering into the device, beside the bolt, nuts and handle. The block is suspended by a bolt through the post and by a chain so as to lie in approximately the same plane as the ropes of the descending and ascending cars.

A ⅝-in. steel-wire rope is used and is attached by means of the safety hook shown in Fig. 261. The cars are in general of two types. That formerly used was built of wood bound with strap iron, and had a vertical front. The door was hinged at the top and was held by a bottom latch, which was tripped by a block beside the track just as the dumping point was reached. These cars were of different sizes, but were rather small, holding about 1000 lb. They have been almost entirely displaced by the car shown in Fig. 261. This car is built rigid throughout and its emptying is facilitated by the inclined end. Its strength is one of its features, as it has to withstand extremely hard usage. Its capacity is about 16 cu. ft. At the upper end are a number of hooks of strap iron arranged to permit coiling excess rope.

Whichever type of car is used, it is dumped by partly upsetting at the foot of the incline, which is the top of the bin; the bin is usually

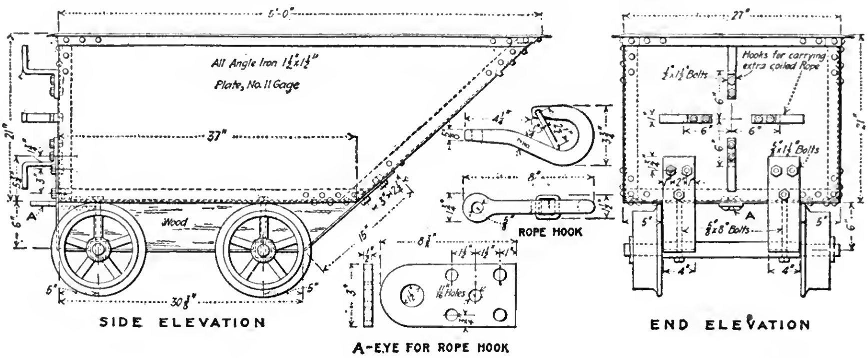


FIG. 261.—GO-DEVIL RIGID CAR AND SAFETY HOOK FOR ROPE ATTACHMENT.

built of wood in a recess blasted out of the foot-wall. This is effected by the device shown in side elevation in Fig. 262. The last 2 ft. of the incline tracks are made steeper and a round timber resting against two inclined posts, as shown, is set across the bottom, over the last tie. The wheels of the dumping car rest on this bumper. In the center of each of the tracks is a hook, usually of 1¾-in. grooved steel; one end is hooked over the next to the last track tie, the other projects upward so as to catch the front axle of the descending car. The axle is caught as the front wheels hit the bumper. The inertia of the car and of its contents causes it to revolve about this axle and to tilt up until the top of the body rests against the crossrail on the inclined posts. In this position the end and bottom of the car are at an angle to permit easy discharge of the contents, aided by the inertia of the descent.

The cycle of operations is as follows: The mucker or trammer fills a car at some point along the lateral tracks; he trams it to the turnsheet

and hooks on the loose end of the go-devil rope; he pushes it carefully over the edge on the track and grabs the brake handle, the empty car at the dump is jerked back so that its rear wheels are again on the rail and as the full car descends, the empty one is pulled up. The length of the rope is adjusted so that, as the full car reaches the dump and empties, the empty car is landed on the turnsheet, whence it is trammed out on the lateral tracks for loading again.

The incline plane itself is kept on as uniform a grade as is convenient. It is built of 4 × 6-in. crossties, about 8 ft. long, spaced 3 ft.; these are

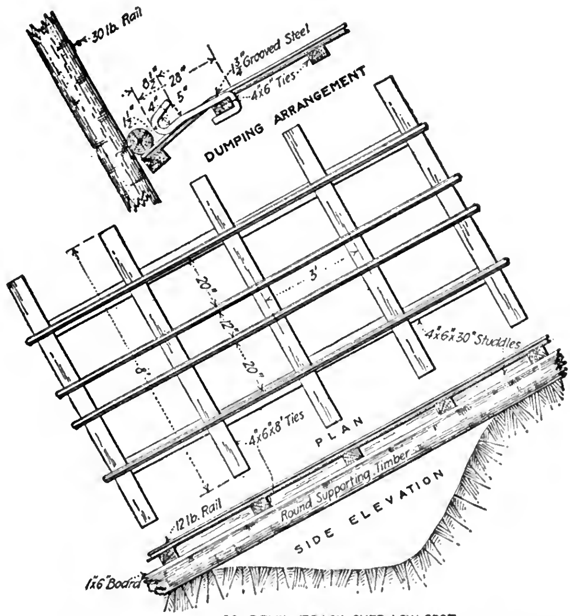


FIG. 262.—ARRANGEMENT OF GRAVITY PLANE AND DUMPING DEVICE.

held apart by studdles of the same material laid on 1 × 6-in. boards; the boards give no support and are used merely for lining in and getting grade. The ties themselves are supported either on the foot-wall or on filling, or on poles across low places. Four 12-lb. rails are spiked to the ties, so as to give a double track from top to bottom with 20-in. gage and 12-in. clearance. The general arrangement is illustrated in Fig. 262.

Improvement in Underground Trolley Conveyors (By E. M. Weston). —One defect of the aërial rope automatic bucket conveyors in use in inclined stopes on the Rand is that unless the haulage ropes be wound on separate clutched drums and have power devices to wind and unwind and thus shorten or lengthen the ropes, the buckets can load from one

place in the stope only. To avoid this, Herbert Krause, underground manager of the New Kleinfontein mine, invented the conveyor shown in Fig. 263. It is really an adaptation of the Whiting hoist principle underground, and is not patented. The illustration explains itself. *X* and *Y* are the main ropes stretched parallel from top to bottom of the slope. *Z* is a swinging stop block to take up any shock. *M* and *N* are two side-tipping, long, shallow trucks of any convenient size, usually 4 to 10 cu. ft. capacity. The haulage rope passes around the two grooved truck wheels *W*, and over a pulley *A*, which is made to traverse along another rope *BC*, so that the working length of the haulage rope in the stope is varied. This gear can work in a stope as low as 45 in. and on a dip of 15° to 35°.

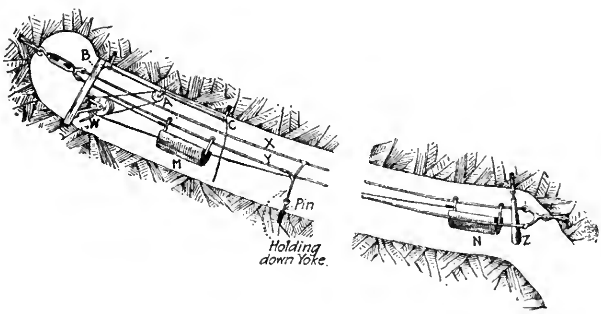


FIG. 263.—ADJUSTABLE OVERHEAD GRAVITY STOPE TRAM.

Hoisting over a Summit (By Frank C. Rork).—The scheme here described was devised to hoist and transport ore from a deposit in the Moose Mountain mine, over a high point, to the mill bins situated about 1500 ft. from the ore. A headframe about 20 ft. high was erected at the highest point between the ore and the bins. Instead of fastening the sheave in the usual way, it was suspended so as to swing and turn according to the pull on the cable. The bail on the skip was constructed so that it would turn over as the car went by the headframe. In the position *A*, Fig. 264, the skip is shown coming up the incline. At *B*, the momentum carries it by the highest point of the track, the bail turning over as it passes. The hoistman then releases the friction and allows the loaded skip to pull the cable out as it goes down the incline to the bins. It is necessary to support the cable on wooden rollers, otherwise the drag of the cable would stop the skip. On the return trip, the momentum of the empty skip is sufficient to carry it past in the same manner. By speeding the hoist on the return trip the round trip can be made in about four minutes, which would limit the capacity to 300 tons per 10-hr. shift, but there is no reason why 4- or 6-ton skips could not be used.

The hoist is belt-driven from a motor (not steam driven as shown in

the illustration). There are two drive pulleys on the motor which are connected by two belts to two pulleys on the hoist countershaft, one 60 in. and one 30 in. in diameter. By means of two friction clutches the 30-in. pulley drives the hoist on the return trip at double the regular hoisting speed.

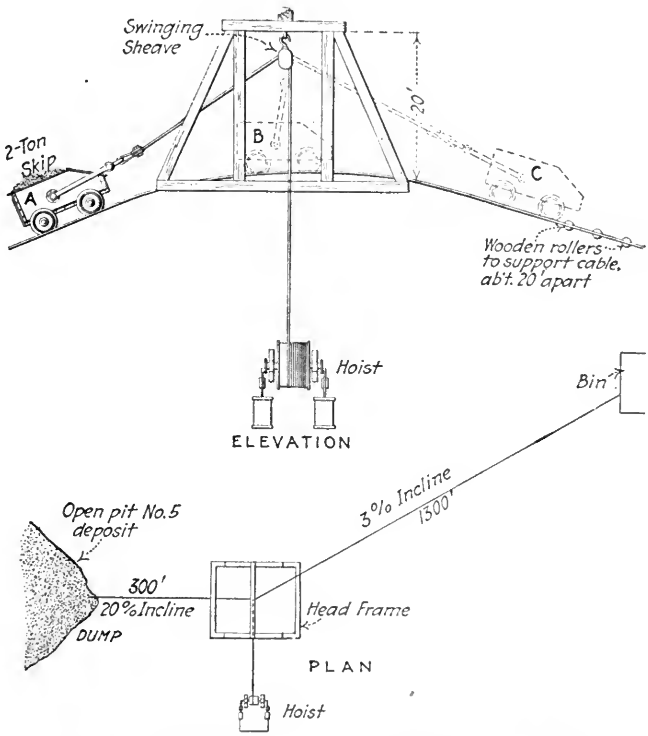


FIG. 264.—ARRANGEMENT FOR HOISTING AND LOWERING.

SIGNALING

Evolution of an Electric Signal System (By George A. Packard).—The Raven mine at Butte was opened originally by prospectors and leasers who very properly followed the vein on its dip with their shaft. Later operators continued this shaft and made so many changes in the angle of inclination that it became difficult to maintain and operate the ordinary type of bell-rope signal. Accordingly it was decided to install electric lights at the stations, using an alternating current of 110 volts, and at the same time to add a third wire and use an electric bell in the hoist room, the bell to be rung by a simple push-button at each station. This worked satisfactorily until the shaft entered wet ground; water then trickled down the wires and got into the push-buttons, resulting in stray

bells in the engine room and confusion to the men hoisting in the shaft. A new management soon replaced the push-buttons on the wet levels with "pull-boxes." This was developed from one in use at the Original mine, and in its present form is shown in 1 and 2, Fig. 265. Changes

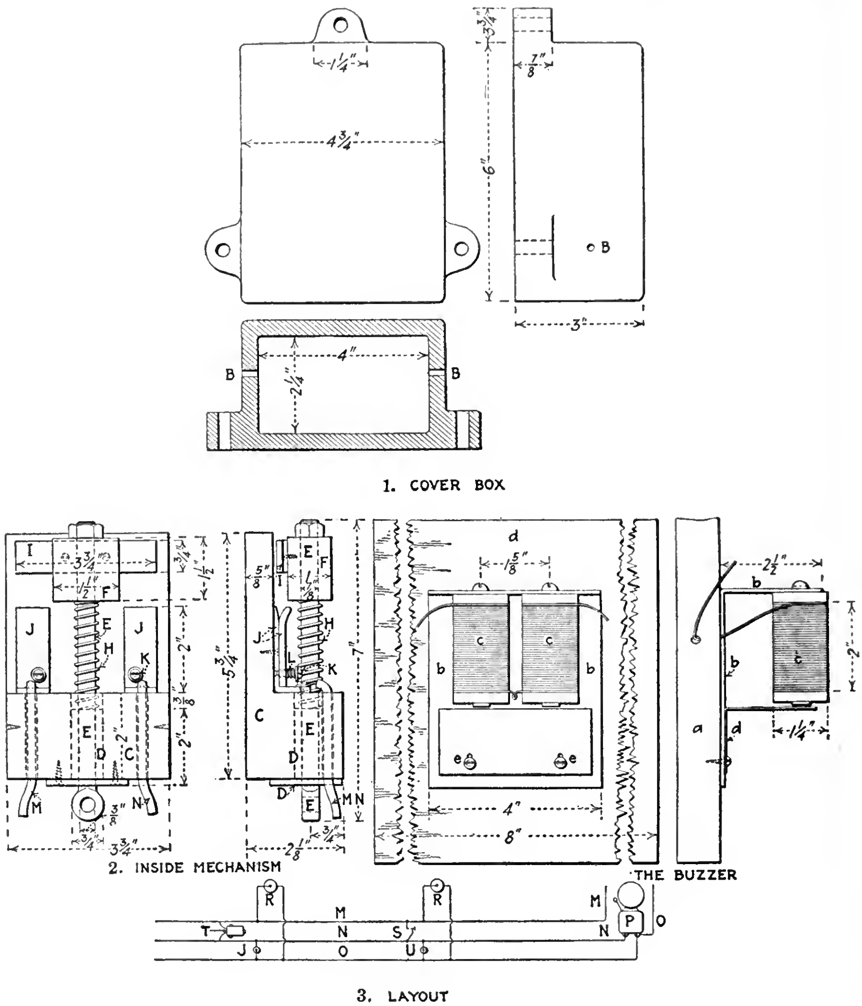


FIG. 265.—PULL-BOX, BUZZER AND LAYOUT FOR ELECTRIC SIGNALING.

were made to eliminate machine-shop labor and to remove the opportunity for stray bells as the box wore.

It consists of a plain cast-iron cover, 1, open only at the bottom to receive the fittings shown in 2. The cover weighs 9 lb., and is heavy enough to stand the blow of any small rock falling down the shaft. It is

fastened to the shaft timbers by long screws or spikes through holes in the three lugs, and has two holes B for holding by screws the wooden block C, which plugs the bottom and acts as a base for the fittings. This block is made small enough to allow for swelling; it is bored vertically $\frac{3}{4}$ in. from the front to admit the cylindrical portion of a mushroom-shaped casting D fastened to the block bottom with screws. This cylinder serves as a guide for a $\frac{3}{8}$-in. eye-bolt E. On the upper end of this bolt a small wooden block F is held by a nut which presses against the top of the box, being forced up by the spring H. The spring is made from No. 12 wire; its bottom rests against the top of the casting D in a socket in C. To the back of the block F is screwed a strip of $\frac{1}{16}$-in. brass I, and at each side of C are fastened four similar strips J, the back two beveled back at the top, and the front two bent forward. The back strips are bent out at right angles at the bottom and the line wires M and N are connected, one on each side, by screws. The front brasses are held in place by the screw K, provided with a washer and a small coiled spring L, $\frac{1}{4}$ in. long. When the bolt E is pulled down, the brass I is forced between the two portions of the brasses J, making contact and, by connecting M and N, ringing the bell. The wires enter the box through two holes C, bored of such size that the insulation fits tightly, and so placed that there can be no possible contact with the screws at B or the casting D. Where there is much "copper water" this casting is of brass. To prevent I from being pulled down so far as to stick, two stops were subsequently screwed into the bottom part of C, between E and J, projecting about an inch above the lower part of C. These are not shown in the illustration. A cord through the eye of the bolt, ending in a porcelain knob, completes the device.

Later, another manager, finding that occasionally a man, ringing for air, say, on the 1100, had no way of knowing that another man was ringing to be hoisted from the 500 at the same moment, while the engineer could not determine what either was after, introduced a flash light into the circuit at each station. In 3 is shown the arrangement at this stage. Wires M and O are those in use for the lighting circuit. The wire N is added for the electric signal system, connecting with P, the bell in the engine room. This bell is of the solenoid type, only one blow being struck each time the button is pushed. The station light is represented by R, and the push-button used on the dry level, by S. When pressed, S closes the circuit between N, otherwise dead, and M, and rings the bell. A pull-box is shown at T similarly placed on the lower levels. The signal lights on each station are represented by U connected between N and O, so as to glow only when N is live. These lights were only 10-watt lamps and the labor of connecting was small. The scheme eliminated interfering signals and worked satisfactorily until an accident to a skip-tender

showed that a simple flash light was not satisfactory, as it was a warning only when watched. Accordingly, a loud buzzer, obtained from the Anaconda company, was substituted, being connected into the waterproof lamp socket.

This buzzer, shown in 4, consists of a piece of board *a*, 8 in. square, to which is screwed a piece of No. 10 sheet iron *b*, 4 in. wide, and 7 in. long, the top 2½ in. being bent out at right angles to form a shelf. Beneath this are fastened two magnets *c* about 2 in. long, wound for 110 volts. These take about 450 ft. of No. 28 wire each. The vibrating member is a piece of No. 20 galvanized iron *d*, 3½ in. wide by 4 in. long, the top 2½ in. being bent over at right angles. This is fastened to the board through slots *e* so that the position may be adjusted, the horizontal portion being about 1/16 in. below the coils. It is planned to vibrate with from 1½ to 2 amp. The whole is covered with a sheet-iron cover to keep it dry. The sound can readily be heard in the shaft 100 ft. away.

The installation of the buzzers led to some trouble with the electric bell in the engine room, as the coils here were wound for 4 amp., but as soon as these were replaced by smaller coils, this trouble was overcome. At the same time the buzzers were installed, a push-button was placed in the line near the engineer's stand so that he could signal to the levels if desired. The possibilty of return signaling in this manner is a most useful and important feature in a signal system and one too often neglected. It offers a means of avoiding mistakes and even accidents.

The wire used in the shaft was formerly No. 10 triple-braid weatherproof, but this was not satisfactory. It seems to give satisfaction where it has a chance to dry out between rains, but where constantly wet underground very soon begins to allow current to leak. The insulation also begins to come off under a constant drip, after several months' service. It was replaced by double-braid solid-okonite wire, and when the shaft was sunk double-braid rubber-covered wire was used. Both have been perfectly satisfactory. The wires are fastened to ordinary porcelain knobs on the hanging side of the manway, and no ducts of any sort are used to protect them.

The only part of the pull-box which cannot easily be made at any mine are the two castings. The total cost of the pull-box at Butte is about \$5. The buzzers cost \$3.15 each. The No. 12 triple-braid weatherproof wire cost \$1.17 per 100 ft. when copper was selling at 17 cts. per pound, while the price of the okonite covered is \$3.16 and the rubber covered \$2.10.

As electric power is supplied to the mines from a substation connected with several generating plants the signal system is rarely out of commission. In a district where the power is liable to be cut off, an auxiliary storage battery would be desirable.

Simple Return Signal System (By H. R. Wass).—The accompanying diagram, Fig. 266, illustrates a homemade electric signaling system installed at the Rosiclare Lead & Fluorspar company's mines at Rosiclare, Ill. It is used for signaling between the shaft station on the ore-hoisting level and the hoist room on the surface. It is simple, reliable, and has so far proved efficient, and it was easily and cheaply installed.

The signal board in the hoist room is about 12 × 18 in. in size, built of hard pine and painted with two coats of P. & B. paint. The switch-

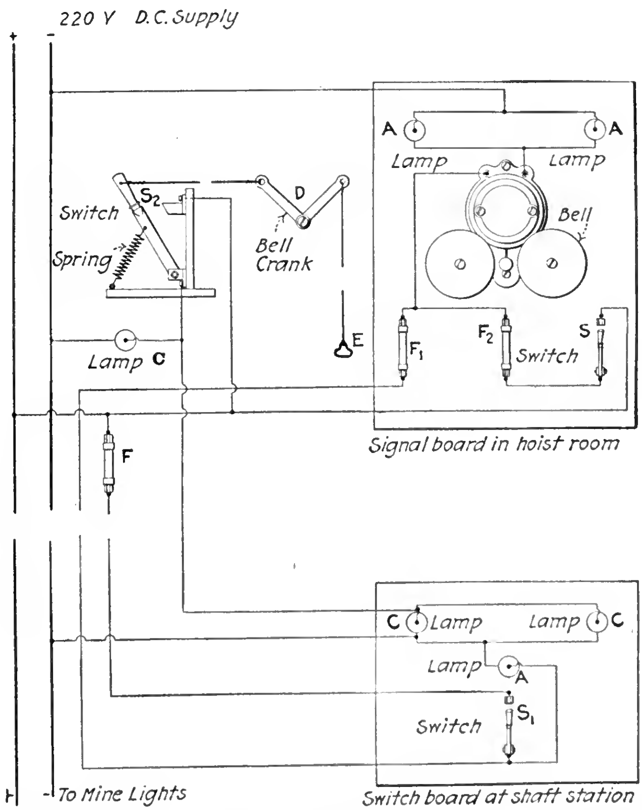


FIG. 266.—LAYOUT PROVIDING FOR RETURN SIGNALS BY ENGINEER.

board at the shaft station is made of the same material and painted in the same manner and is protected from dripping water by a wooden canopy. All wiring is double-braid rubber-covered and is carried down the shaft, on the mine level and in the shaft station, in metal conduit and is fully protected from mechanical injury.

The method of operation is as follows: To hoist, the cager closes the switch *S*1, a single-pole, single-throw, knife switch, and holds it closed

for two or three seconds; this lights the lamps *A* and rings the bell in the hoist room. Two of the lamps *A* are in series with the bell to furnish the necessary resistance. The hoisting engineer responds by pulling down on *E*, which closes the switch *S*2, also a single-pole, single-throw, knife switch, which is held open normally by the spring; this operation causes the lamps marked *C* to light up and remain lighted until the spring pulls the switch open. Two of these lamps are on the switchboard in the shaft station and the other is located conveniently to the engineer's operating platform so that he may observe whether or not the cager receives his return signal. All of the signals are repeated by the hoisting engineer after the underground signals are received by him and all misunderstanding of signals is thus avoided. *D* is a simple bell crank forged from $\frac{1}{4} \times 1$-in. bar and connected to the handle of the switch by a small rope. The switch is mounted on a wooden frame as shown, or may be placed in any other convenient position.

At *F* and *F*1 are 5-amp. fuses. Whenever the engineer or electrician wishes to test or adjust the bells, he simply removes the fuse from *F*1, places it in the socket at *F*2 and closes the switch *S* which rings the bell and lights the lamps *A* on the hoist-room switchboard. All lamps are carbon filament, 50-watt, 220-volt lamps working on a 220-volt direct current. The bell is of a weatherproof ironclad type, wound to a resistance of 30 ohms; it consumes 0.3 amp. of current.

Electric Signal System, Argonaut Mine (By R. S. Rainsford).—The electric signal apparatus here described was devised to meet the following requirements: (1) To be available from a moving skip or cage, as well as from a station platform; (2) to be easily rung, as by a light pull on the bell signal cord; (3) to be absolutely reliable, even if portions of the signal cord in the shaft were broken; (4) to register one stroke of the electric bell for each tug on the cord. For reasons to be explained this last requirement is the crux of the whole problem.

The great depth of the Argonaut shaft, and the fact that it is an incline (60°) made the ringing of signals by the ordinary bell cord a difficult task. For, despite the addition of long hand levers for ringing and the supporting of the signal cord at intervals by sensitive S-shaped springs, together with the greasing of all cleats guiding the cord and constant supervision of the line, the system was anything but satisfactory. Consequently when telephones were introduced in the mine a few years ago a four-wire cable was laid in the shaft, two of the wires being used for the telephone circuit and two for a system of electric push-buttons situated at all stations and loading chutes in the shaft. These push-buttons, greatly modified in design, are still in use, but have the great disadvantage that they are not available from a moving skip.

The most common method of meeting the first requirement above

mentioned, namely, two naked wires strung in each hoisting compartment about 4 in. apart, and charged with current of exceedingly low potential, is usually unsatisfactory and dangerous. To ring a bell one has only to short-circuit the two wires with any convenient piece of metal, as a candlestick. But two objections present themselves. One is the rapid eating away of any uninsulated wires, whether galvanized iron or copper, when subjected to the mine waters and atmosphere of certain mines. The other is the danger of a signal given by mistake through contact with the wires by any tool in the hands of a miner repairing the shaft, particularly a bar when the skip tender is barring a loading chute in the shaft.

Another system was tried which included bonding the shaft rail for one leg of the current, insulating the skip and using the hoisting cable for the other leg or conductor, together with a signaling device on the skip to make or break the circuit. This system proved exceedingly troublesome, either on closed or open circuit, and was finally abandoned. The only solution remaining therefore was some yet undetermined method involving a more sensitive signal cord in connection with the electric gong of the push-button system and its cable.

To make the signal cord sensitive was an easy matter. It was cut in 500-ft. sections, the bottom of each section fastened to a divider of the shaft sets, the top of each section supported by an S-spring, as shown at *F* in 4, Fig. 267, and the signal cord guided by cleats in the usual way. Then a bell crank, with 8-in. arms, was connected to the signal cord below the S-spring and its free upper arm used through an appropriate device to register every pull on the signal cord. A signal cord has one peculiarity —all pulls on it are not of equal strength. One may easily arrange an electric contact which will be made or broken, depending on whether open or closed circuit is used, by a light tug on the cord. The problem is to prevent a double signal if this sensitive cord is pulled so hard that the contact is passed and the mechanism must retrace its steps and again pass the ringing contact as the signal cord is released. If a long sliding contact is used to meet this difficulty the electro-mechanical gong at the surface may have time to ring twice before the contact is broken. The solution in the case of the signal device to be described is to jump the contact on the return stroke, or release of the signal cord.

Thus a light pull on the signal cord drives the lifting member *S*, 1 and 2, Fig. 267, under the lifter pin *R* and so closes the contacts *Q* and *O* by raising *R* and the spring *P*. On the release of the signal cord, *S* drops back to its normal position in front of the stop or guide *Y*. A heavier pull, however, drives *S* under *R* and out beyond, thus both making and breaking the contact at *Q* and *O*, but on the release of the signal cord *S*, which has a laterally projecting portion beveled on the under

side as well as on the top, as shown in 3, slides over instead of under *R* and so makes no contact on the return stroke.

The rest of the apparatus is easily understood from the figures, in which *K* is a wooden base, *L* a slate slab, *Z* brass connectors with binding posts *N* for the wire terminals *M* of the electric circuit. On the two brass connectors are mounted the springs *O* and *P*, the latter supporting also the contact point *Q* and the lifter pin *R*. A bracket *W* supports the pin *R* at the proper elevation, while another bracket *X* serves as a guide and backstop *Y* for the lifting member *S*. The latter is fitted with the

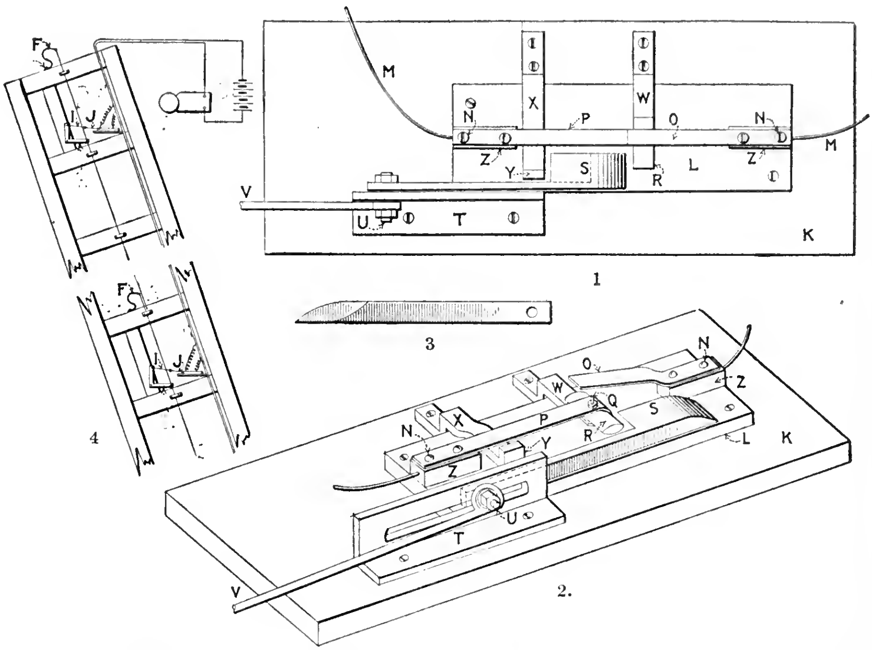


FIG. 267.—LAYOUT AND CIRCUIT CLOSER FOR COMBINATION CORD-ELECTRIC SIGNAL SYSTEM AT ARGONAUT MINE.

stud *U* running in the guide *T*. To the stud is fastened the link or rod *V* leading to the upper arm of the bell crank *I* through the insulating block *J*.

The device is applicable to any shaft whether vertical or inclined, and if any section of the signal cord should break, or any individual circuit closer be damaged, signals in other sections of the shaft are not interfered with. In practice, moreover, it is found that a signal can easily be rung from a fast-moving skip by giving the signal cord a jerk with only one or two fingers.

Bare-wire Electric Signal System (*Bull.*, American Institute of Mining Engineers).—The Penn Iron Mining Co. in Michigan uses a

rather unusual system of electric signaling. An alternating current of 110 volts taken from the lighting system is reduced to 30 volts through a small transformer. This 30-volt circuit is used to ring what is called a grade bell, *i.e.*, a signal to indicate the class of ore hoisted, and also to operate two relays, one of which rings the skip bell in the hoist house by closing a 100-volt circuit, while the other similarly rings the cage bell. One side of the 30-volt circuit is grounded, the other connected to the grade bell and the relays as shown in Fig. 268. The other sides of the grade bell and the relays are connected to No. 4 bare copper wires supported on insulators down the shaft. By grounding any one of these copper wires, a current will flow through the grade bell or a relay and give a signal. The bell wire for the cage is near the center of one side of the compartment so that it is almost impossible to reach and ring it unless one stands on the cage. This prevents ringing the cage bell when the

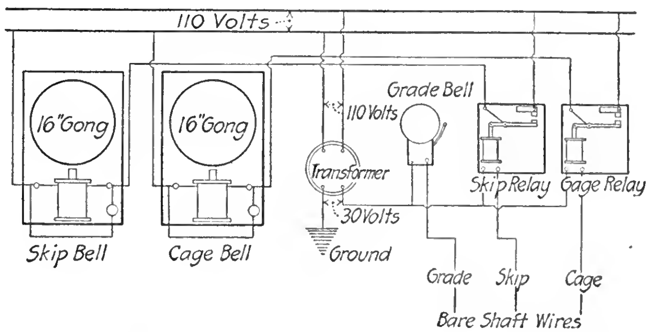


FIG. 268.—CONNECTIONS FOR RELAY AND DIRECT SIGNALING.

cage is not in sight, such ringing being a frequent source of accidents. A short piece of flexible wire with a piece of bare No. 4 wire on its end is fastened to the iron work of the cage so that the shaft signal wire can be grounded from any point in the shaft, whether the cage be moving or at rest. The skip and cage bells are 16 in. in diameter and are set in a local 110-volt circuit, which is closed and opened by the relays mentioned. The blow on the gong is struck by a bar in a heavy solenoid. An indicator which registers the number of bells rung is also actuated and a lamp is lighted.

Gravity-release Electric Signal Box (By W. R. Hodge).—An electric-contact signal box which has proved satisfactory through several years of hard service is shown in Fig. 269. The device consists of a fixed block nailed to the top of an inclosing box and a movable handle extending through the end of the box. The block and handle are connected by a butt hinge. Brass plates for making contact are screwed to both the block and the handle. Soldered to the side of each brass plate is a wire

connecting with the bell line. The handle must be lifted to make a contact and give a signal. The contact made, the handle naturally drops to the open position without further attention. Ordinarily the boxes are unpainted but in unusually wet places a coat of paint and a shield which moves with the handle and covers the handle opening into the box, make for increased efficiency. These boxes have worked well in wet stations.

Locked Signal System (By H. H. Hodgkinson).—The signal system installed in the Palmer shaft of the New Jersey Zinc Co.'s mine at Franklin, N. J., by R. M. Catlin and designed by L. G. Rowland is unique and has many decided advantages over other signal devices. A signal box is located at each shaft station and is operated by a lever on the side, to which is attached a short chain and handle to facilitate pulling the lever. The cage, however, is moved only on signal given to the engineer by a man who carries a key to the device and who travels with the cage. The signal boxes are situated at each station in such a position

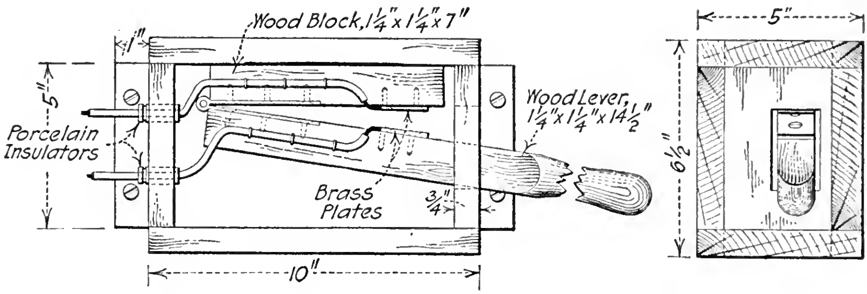


FIG. 269.—CONTACT IN BOX, CLOSED BY LIFTING HANDLE.

as to be easily accessible to the cage conductor and enable him to see his cage constantly, thus minimizing the danger of mishaps from the cage moving while men are getting on or off.

In operating, the pull and release of the lever ring the gongs in all other signal boxes, but do not ring the engineer. Thus it is always possible to signal for the cage from any station in the shaft and to notify anyone about to ring a signal from any other station that there is already a signal being rung, thus preventing confusion. As soon as the cage conductor receives a signal—provided there is nothing to prevent him from bringing the cage to the station from which the signal was rung—he inserts a special key in the lock on the front of the box and turns it, thus connecting the one signal box with the engineer's signal in the engine house. He then signals the engineer by means of the requisite number of pulls on the lever; this signal at the same time rings at each station, notifying the persons who rang the original signal that the cage is coming. The conductor remains at the signal box with his key in the

lock waiting for the engineer to repeat his signal. If the engineer repeats correctly, the conductor removes his key, climbs on the cage and proceeds to the station signaling. If, however, the signal rung back by the engineer happens to be wrong, the conductor rings one bell to the engineer to stop and the exchange of signals is repeated. There is always ample time allowed the cage man to remove the key and climb on the cage.

Fig. 270 shows a diagram of the system. This consists of the two circuits Y and X. Through circuit X there is a constant current flowing which causes the electro-magnets to operate the clappers of the gongs in the signal boxes when the pull switch is thrown. It will be seen that

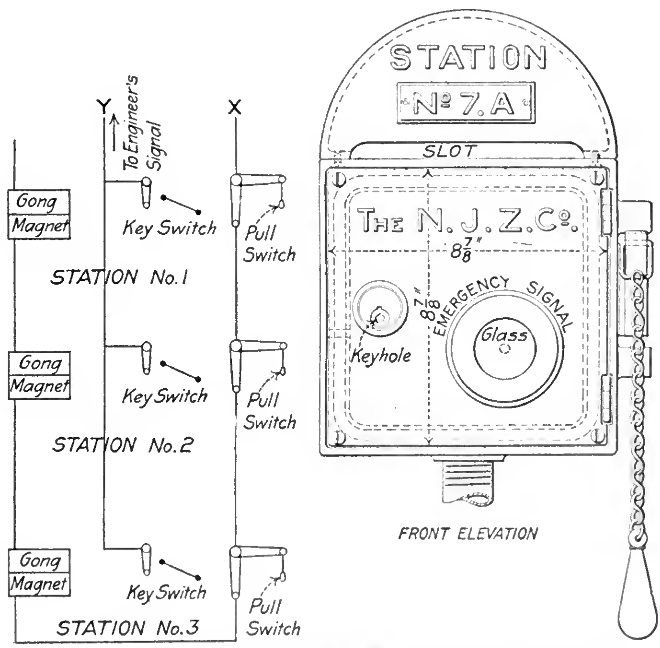


FIG. 270.—DIAGRAM OF SYSTEM AND FRONT OF SIGNAL BOX.

this operation does not operate the engineer's signal. Through the circuit Y there is no current, and when the key switch is thrown the signal box is merely connected to the engineer's signal, which cannot ring until the pull switch is also thrown, causing the current to pass through circuit Y and operate the engineer's signal.

The signal boxes are constructed of cast iron as shown. The front of the signal box, Fig. 270, is in the form of a door to permit repairs to be made inside. It is locked by means of the same key which operates the engineer's signal, in addition to being fastened by a screw in each corner. The doors are fitted on the inside with a $\frac{3}{8}$-in. round-rubber

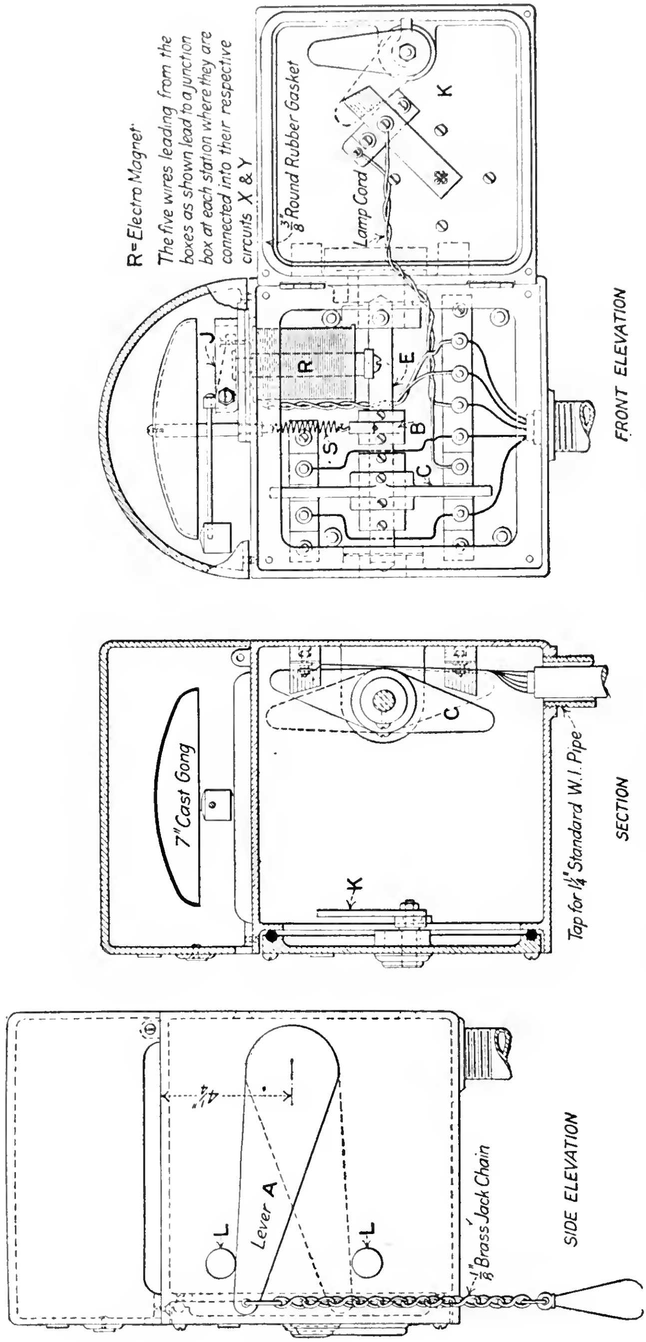


FIG. 271.—SIDE ELEVATION AND INTERIOR OF SIGNAL BOX.

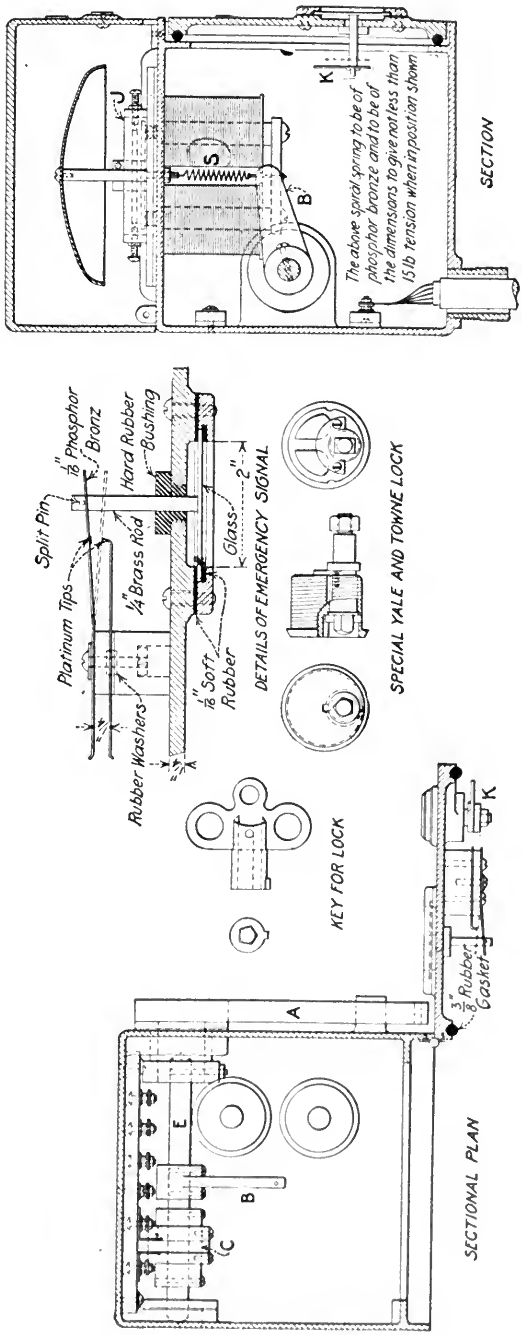


FIG. 272.—DETAILS OF SIGNAL BOX AND SAFETY LOCK CONNECTIONS.

gasket to prevent water from getting into the boxes. The locks are so constructed that it is impossible to remove the key and leave the key switch thrown and the signal box connected to the engineer's signal. On the door of the signal box an emergency signal device is placed so that the cage can be stopped immediately in case of an actual or threatened accident. This consists of a brass pin held in place by a round piece of window glass, which when broken permits the pin to be forced outward by means of a spring, making the same connection as the key switch would. The lever is then pulled once, which will bring the cage or skip to a stop if in motion. The gong is placed on top of the signal box proper and over it is bolted a tightly fitting cast-iron cover, which prevents any water from working its way inside. The cover has a ½-in. slot on the front and sides to enable one to hear the gong more distinctly.

The lever *A*, Figs. 271 and 272, which operates the pull switch is situated on the outside of the box and has its movement confined to the space between the two lugs *L L* to prevent any injury to the switches inside. The pull-switch lever *C*, in the position shown in Fig. 271, permits the current to pass to the electro-magnet which holds the clapper up against the gong. When the lever *A* is pulled down, the lever *C* assumes the position shown by the dotted lines, which breaks the circuit *X*, causing the clapper to drop and at the same time closes the circuit *Y*, when the key switch *K* is thrown, and thus rings the engineer's signal. Upon releasing the lever *A* it is returned to its normal position by the spring *S*, Fig. 272, the circuit *X* is closed again, causing the clapper to strike the gong, and the circuit *Y* is broken as before. The spring *S* is fastened to the short lever *B*, which operates the shafting *E*, to which the lever *A* and the switch lever *C* are attached.

Bell-wire Arrangement in Sinking (By Clinton P. Bernard).—In shaft sinking it is necessary to carry down the signal-bell wire or rope as depth is gained. In shafts over 200 to 300 ft. deep, it is common to use a heavy wire or wire rope from the surface to a point about 50 ft. from the bottom, and a rope for the rest of the way. There are numerous ways of arranging the wire and bell at the surface, but the one shown in Fig. 273 is simple and has many advantages. The wire is wrapped three or four times over a wooden spool attached to the post of the headframe, then passed through the handle of the weight cylinder, lashed, and the rest of the coil hung in some convenient place. The weight cylinder is a piece of 2½-in. pipe with a cap on the bottom, filled with scrap to give the desired counterweight. This weight cylinder moves in a 4-in. pipe outside the collar set and extending, for convenience, about 3 ft. above the ground. A bumper on the spool permits it to revolve just enough to actuate the bell through a wire fastened to an arm

on the spool. The shaft wire may be lengthened as desired, by unfastening the lashing above the weight cylinder, and allowing the required length of wire to slip over the spool, thus doing away with splicing. No springs are required and there are no weights to dislodge and fall into the shaft. By adjusting the counterweight, the pull on the wire need be but a few pounds through 3 or 4 in. The entire outfit can be made in a short time, at small expense.

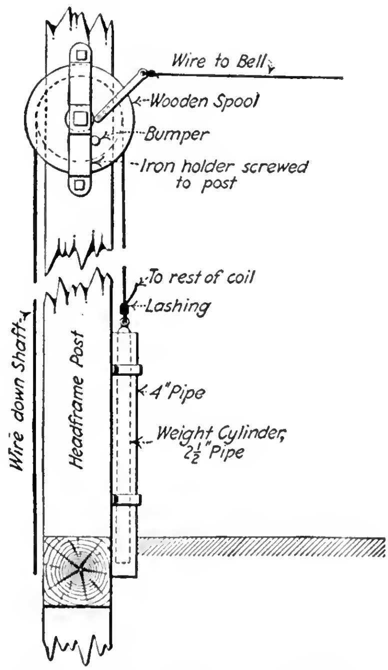


FIG. 273.—ARRANGEMENT OF SIGNAL PULL-WIRE AT SHAFT COLLAR.

Warning Bell for Topman (By Harold A. Linke).—At many small shafts the "topman," besides acting as skip tender and car "rustler," is engaged in making ladders or striking for the blacksmith, etc. These odd jobs take him away from the shaft, and even though he be exceedingly careful and conscientious, he is likely to miss a bucket occasionally. As the rule is not to delay shaft work, any contrivance which tends to obviate such delay makes for efficiency. Fig. 274 shows a simple device for completing an electric-bell circuit, for use on the indicator of the hoist. A depth- or station-marker, furnished with the indicator, can be fastened at the position desired by means of a setscrew. A piece of galvanized iron is bent as shown and punched with a hole large enough to pass the setscrew without contact. Two mica insulators, such as are furnished with ignitors of gas engines, or thin rubber gaskets are used to set the

galvanized iron off from the setscrew and the marker. A cut metal washer is applied as shown. A copper wire to the bell is soldered to the galvanized iron; the other wire may be grounded or fastened to some part of the engine, such as a screw of the name plate. The marker is set at some point on the indicator so that, in hoisting, the contact between the movable pointer and the piece of galvanized iron may be made, and the warning bell rung sufficiently early for the topman to be at the shaft collar to meet the ascending bucket.

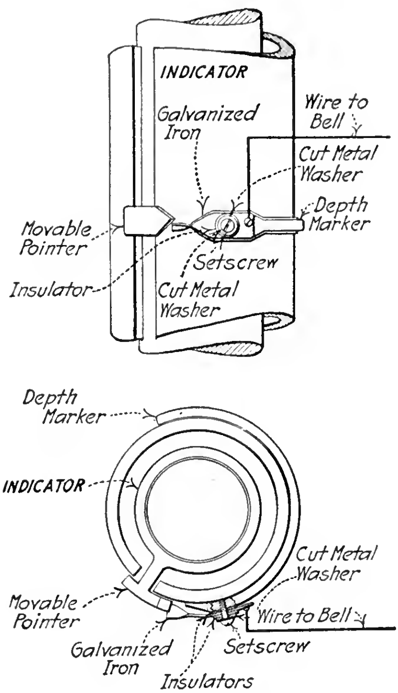


FIG. 274.—DEVICE FOR COMPLETING ELECTRIC CIRCUIT OF WARNING BELL.

Automatic Light Switch for Electric Tramming (By L. O. Kellogg).—It frequently happens in underground electric-haulage systems that the main drift branches a short distance from the shaft and that loaded trains are brought in over both branches. It then becomes necessary to signal to any incoming trains whether the track between the junction and the shaft is open or whether a train from the other branch drift is dumping at the pocket. This signaling is done automatically by colored lights in a neat way at the Oliver mines near Ely, Minn. A switch is set in the back of the drift so as to be tripped by the trolley wheel of the incoming train and to flash a red light. The trolley wheel of the outgoing train usually flashes a green light. A train approaching on the other

branch drift will stop if the motorman sees a red light and will proceed if he sees a green. It is not entirely necessary to use the green light, but somewhat safer; for if two lights are used, the absence of either a green or a red light indicates that something is wrong with the system, and the motorman will not proceed as he would if the mere absence of a red light was used to indicate a clear track. Only one switch is necessary, set where it will be operated by trains from both drifts. One or two sets of lights may be used, only one set being required if a point can be had visible from a sufficient distance in both drifts; this is sometimes impossible on account of the curve arrangements.

The construction of the switch is shown in Fig. 275. It consists of a wooden box inclosing the contacts. A vertical pivoted finger is connected through the bottom of the box with two horizontal wings which the trolley wheel strikes; the finger, thrown laterally by the movement of the wings,

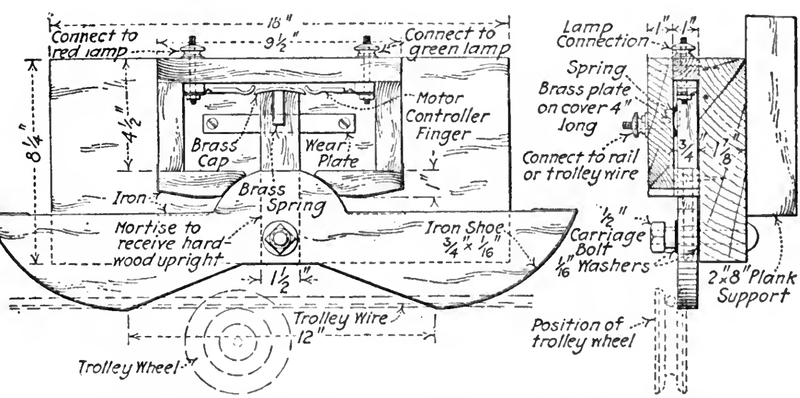


FIG. 275.—SWITCH TO SIGNAL POSITION OF UNDERGROUND MOTOR.

completes a circuit through the lamp of the proper color. The wings of wood are shod with iron where struck by the wheel and where they strike the bottom of the box, in order to resist wear. The top of the wing-piece is shaped as an arc to fit the opening in the box, which is thus kept closed. The box itself consists of a wooden back and a wooden front piece screwed together through filling blocks which form the box sides. The finger, by means of the brass spring shown, maintains a constant contact with a brass plate on the inside of the box front piece, which is connected to one side of both lamp circuits. The spring connects with a brass cap on the finger, and this makes contact with one of two fingers taken from a motor controller. One of these connects with the other side of the red-light circuit, the other with the other side of the green light. In the case shown, the trolley current is used for the lights; if a separate lighting circuit were in service, connections could be as easily made to this.

This device is mounted on a board fastened parallel with the tracks between two caps.

WINDLASS AND WHIM

Test-pit Windlass (By L. D. Davenport).—In the openpit mines of the Chisholm district, on the Mesabi range, it is the custom during the winter season to put down a large number of test pits in the ore for exploration purposes. These pits are usually about 3 × 4 ft. in cross-section and range in depth from 10 ft. to 80 ft. Fig. 276 shows the details of the windlass used to hoist the ore from these pits. Four railroad ties are placed around the mouth of the test pit to form a collar and the bottom of the windlass frame is spiked to this collar. The windlasses used for underground test pits are made on the same general plan, except that they are smaller.

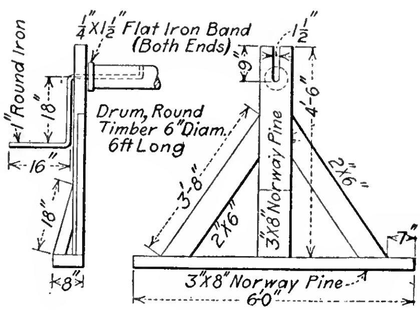


FIG. 276.—A SIMPLE HAND WINDLASS.

Windlass for Single-hand Sinking (By Albert G. Wolf).—The accompanying illustrations, Fig. 277, show a windlass and a self-dumping mechanism with which a prospector can sink a shaft alone, while saving the labor of climbing in and out of the shaft to hoist each bucket. The windlass is small enough to be placed at the bottom of a shaft and light enough to be installed, operated and removed by one man.

The windlass, as shown in plan and elevations, consists of a wooden drum, 1 ft. long by 6 in. in diameter, an axle and crank made of a single piece of round iron, and a yoke of round iron, which supports the drum and crank. The yoke is held by two eye-bolts fastened through a piece of 2 × 6-in. plank. The windlass is braced and prevented from swinging in the eye-bolts by a third piece of round iron, one end of which is bent around the axle at the crank end of the drum, and the other, hook-shaped, passed through a third eye-bolt in a horizontal piece of 2 × 12-in. plank. The two planks are spiked securely together. When the windlass is to be set up, the vertical piece is placed at the center of one end of the shaft and the horizontal piece is wedged firmly between the

two walls, just as a stull would be. The length of the horizontal piece will be varied according to the size of the shaft being sunk. As the drum is small, space for extra cable is made by driving two rows of pins, about an inch apart, around the drum near one end. These pins are large nails which have the heads cut off.

The framework of the dumping device is made entirely of 2 × 4-in. lumber. The bucket slides on two skids to the top of the shaft. Here, two lugs, riveted to the bucket below its center of gravity, engage two outer skids, to which are fastened two beveled pieces; the bucket travels up these as on an incline plane. When the top is reached, the lugs strike two pivoted pins and drop over the ends of the planks. By slacking

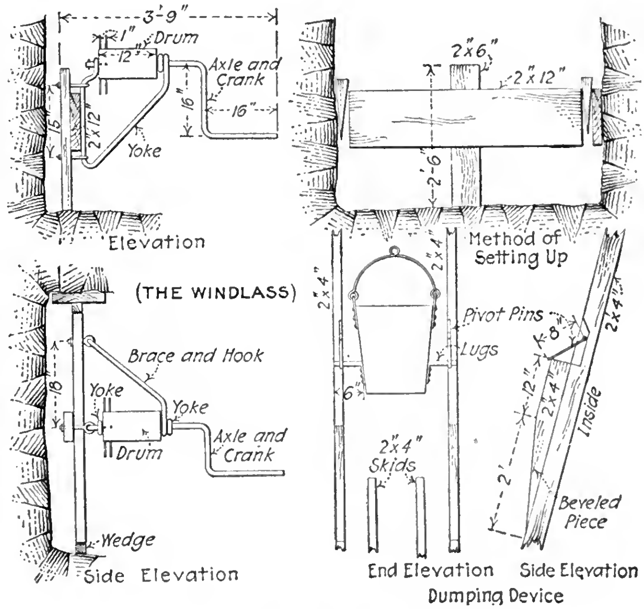


FIG. 277.—ONE-MAN SINKING LAYOUT.

the rope at this point, the bucket is allowed to turn over on its lugs and dump its contents into a chute properly placed. The bucket is then hoisted a few inches over the pins, which fall into place, and the lugs guide the bucket over them to the skids again. When it is desired to remove the windlass and protect it from blasts, enough muck is placed in the bucket to balance the windlass, and the bucket is hoisted to twice the height that it is desired to raise the windlass. The hook brace is then loosened, the drum is swung against the frame and fastened, the wedges are knocked out and the machine ascends without much effort on the part of the operator. This machine was devised by W. J. Finney, of Luning, Nev.

Device for Handling Prospecting Bucket (By Thomas M. Smither).—The device illustrated in Fig. 278 is designed to allow extending the dump from a prospecting shaft without arduous handling of the bucket. The action of the contrivance is apparent from the drawing. The hoisted bucket is released and placed on the movable end of the board, which is then revolved about its pivoted end by sliding on the track *A* until the dumping point is reached. The fixed end of the 2 × 12-in. by 14-ft. revolving board is bolted to a 12 × 12-in. block with washers inserted to insure easy motion. The 1-in. round iron *A* is an old cyanide tank hoop. It is fastened to the supporting boards by nails on the sides, the heads of which are kept below its top. Pipe or rail would answer the purpose as well. The bottom of the board was protected against wear on the moving end by iron straps. The iron was greased and the loaded board easily moved thereon.

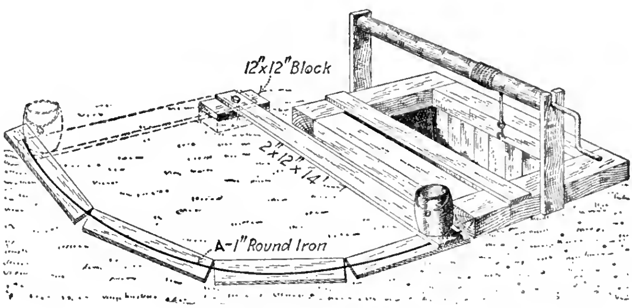


FIG. 278.—PROSPECT BUCKET DUMPING ARRANGEMENT.

Lowering in Balance through Timber Shaft.—In the iron mines of the Mesabi range the timber and boards used underground are seldom handled in the main hoisting shafts. Instead, one or more timber shafts are sunk at different points, affording ventilation and saving underground tramming. There are several different styles of timber shaft ranging from the inclined slide to the more elaborate type with counterweighted cage. Fig. 279 shows a simple headframe used on the timber shafts of one of the larger mines. The 1⅛-in. manila rope is passed five or six times around the drum and made of such a length that one end is at the bottom of the shaft when the other end is at the top. There is, therefore, no labor wasted in raising the empty rope as there is in most timber shafts. Two men work together, one operating the drum and the other handling the timber or boards. The headframe and drum are constructed by the mine carpenter and assistants in from two to two and one-half days. The length of the drum should be carefully determined in proportion to the depth of the shaft so as to prevent a lateral travel of the rope greater than the length of the drum. One of these headframes is now being used to lower timbers 160 ft., with perfect success.

Lowering Windlass for Timber Shaft (By L. D. Davenport).—The windlass shown in Fig. 280 was constructed for the purpose of lowering timber and other material down a shaft. The drum is made of a piece of iron pipe 14 in. in diameter outside and 4 ft. 6 in. long. This is connected to a 2½-in. square axle by means of two iron castings, riveted to the pipe with ⅜-in. countersunk rivets. One of these castings is larger than the other and forms the brake wheel *A*. It is 28 in. in diameter with a 6-in. face and has a 2-in. shoulder turned to fit the inside of the pipe. The axle is made of 2½-in. square stock and is 7 ft. 2 in. long over all. The last 2 in. on each end *B* is made 1½ in. square to receive

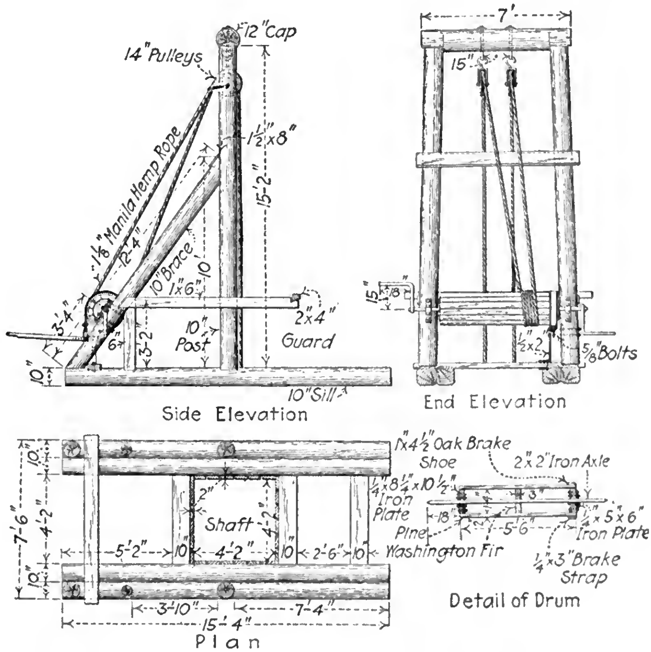


FIG. 279.—WINDLASS AND HEADFRAME FOR LOWERING IN BALANCE.

cranks used for turning the windlass; the next 8½ in. on each end is turned to 2½ in. in diameter for the bearings, which are 2½ × 7½ in., and babbitted ¼ in.; and 5 ft. 5 in. is left in the center of the axle 2½ in. square. The brake band is of 3/16 × 6-in. iron, 6 ft. 8 in. long, and is drilled with 9/16-in. holes to allow eight basswood brake-block segments 2 in. thick, to be bolted to it. These basswood blocks last from eight months to one year. The ordinary fulcrum, brake-band connections, and turnbuckle are used, as shown, and for applying the brake a ¾ × 3-in. lever 6 ft. long. A semicircular sheet-iron cover is placed over the brake wheel to protect it from the weather.

A $\frac{3}{8}$-in. or $\frac{1}{2}$-in. wire tiller-rope is used in the shaft with a 20-ft. length of chain on the end. In operation the timber truck is run up to the collar of the shaft and two half-hitches are taken around the timber or lagging with the chain. One of the landers then raises the load by turning the drum with a cant hook in the 1-in. holes near one end, until the load slides off the truck into the shaft, the second lander meanwhile holding the weight by means of the brake lever. The load is steadied by the first lander and is then lowered to the bottom. Steel tram cars, weighing approximately $1\frac{1}{2}$ tons, have been lowered easily and slowly with one of these windlasses. It is intended to arrange a set of gears, similar to an ordinary hand winch, on the brake-wheel end of the drum, so that a load can be raised off the timber truck without using the cant hook. For if the load drops even a very short distance, the lander us-

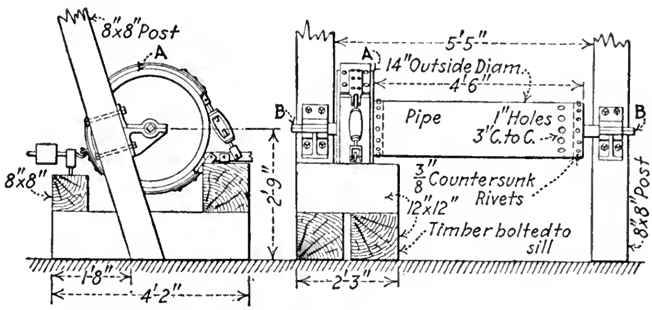


FIG. 280.—IRON WINDLASS FOR LOWERING TIMBER.

ing the cant hook is liable to be hurt. The first of these iron windlasses was made in 1910 in the shops of the Oliver Iron Mining Co., and was used in the Chisholm mine. They have proved highly satisfactory for the service demanded.

Joplin Type of Horse Whim.—Missouri horse whims are built on the idea of placing the drum and brakes at the collar of the shaft, leaving the control of the horse to word of mouth. This entails no danger from the horse or mule becoming unmanageable, as the driving shaft can easily be thrown out of gear with the drum and the bucket held in the shaft by the brake lever, the same lever being used for both operations. The headframe used with these hoists is made high so that where the ground is flat an upper floor may be built, which is necessary in order to get dumping room. On that account the brake lever, a piece of 3 × 3-in. timber, is made long enough for the driver to control the hoist from the upper floor while landing the bucket.

The design of the whim is shown in Fig. 281. The horsepower gear is so placed that a 14-ft. tumbling rod will connect with the drum shaft. The horse is attached to a sweep, 12 to 14 ft. long, and as there are

77 teeth on the driving gear and 12 on the pinion, and as the drum is a little more than 10 in. in diameter, the bucket is raised about 17 ft. to one journey of the horse around the ring. A knuckle is used to connect the tumbling rod to the shaft of the power pinion as well as to the shaft of the hoisting drum.

The drum is carried on a 4 × 4-in. oak frame, as is also the driving gear. The drum frame is securely fastened to the frame of the derrick in an upright position. The rope, which is generally a discarded 5/8-in. rope from a steam hoist, reels directly on the drum from the top sheave. To prevent the bucket from running back when the drum is in clutch, a ratchet and dog are provided. These are shown in the illustration and are marked *A* and *B*. The drum is loosely mounted on the drum shaft, while keyed to the driving shaft is a double-arm dog *C*, which engages with

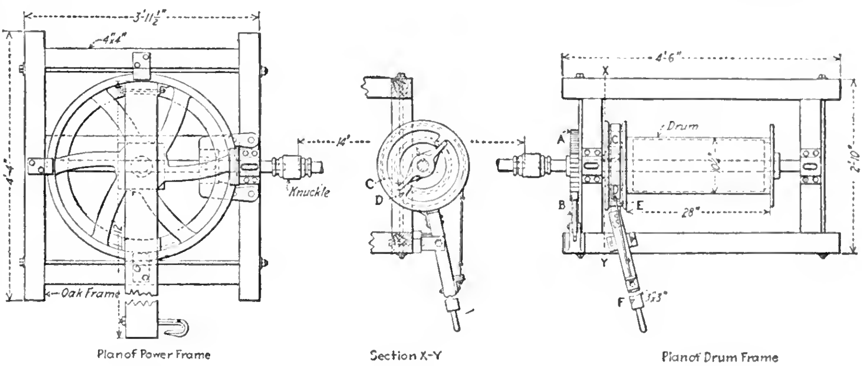


FIG. 281.—DRIVE GEAR AND HOISTING DRUM OF JOPLIN FRAME.

the two plugs *D*, forming part of the end casting of the drum. An outer groove is made in this casting for the brake strap *E*, which is lined with leather, since wood is too bulky for brake lining on such a hoist. Pieces of old belt are used for this purpose; it lasts well, and gives a good grip on the drum. The brake lever is fulcrumed either on the top cross-brace of the drum frame or else on a 2 × 6-in. crosspiece nailed to the headframe about 18 in. from its lower end. This is a loose fastening permitting side swing to the lever, as it is by the side throw of the brake lever that the drum is thrown into or out of clutch with the dogs on the driving shaft.

ROLLERS AND SHEAVES

Rope Idlers for Incline Shaft (*Bull.*, American Institute of Mining Engineers).—The shaft of the Raven mine, Butte, Mont., is an incline 1700 ft. long and with various dips. At the top the dip is 70° and gradually flattens until at the 300-ft. level it is only 47°. This dip continues

to the 1100-ft. level, below which it curves with a 125-ft. radius to 78°. The shaft, furthermore, does not lie in one vertical plane, so that the hoisting rope not only rubs at intervals on both the hanging and foot-walls, but presses strongly against the west dividers near the collar, while 300 ft. below, it runs close to the east end-plates.

The early operators used no idlers, and wall plates cut 6 in. deep by the rope resulted. Later operators first attempted to overcome the excessive friction, and the wear of rope and wall plates, by introducing solid cast-iron idlers, 3 in. in diameter. To allow for the travel of the rope from side to side, some of these had to be 3 ft. long and were extremely heavy. Judging from the appearance of the old idlers of this type found at the mine, they often failed to turn in the bearings, which is not surprising when it is considered that they would have to make 1000 r.p.m. under ordinary hoisting conditions.

The next rolls were of wood, 6 in. in diameter, with an iron band about each end, and a pintle of 1-in. round steel driven in at the ends to serve as a shaft. These wore rapidly, and were soon replaced by rolls made from water pipe, 5 or 6 in. in diameter, cut to the desired length and fitted with a wooden cylinder into which the pintle was driven. Where the idlers were used on the hanging wall of the shaft the original bearing was simply a piece of $\frac{1}{2} \times 1\frac{1}{2}$-in. strap iron, 10 in. long, turned up at the end in a circle of $1\frac{1}{8}$-in. diameter. A small hole served for oiling, and common black oil or filtered oil from engine bearings and compressor bearings was used. When the rolls were to be placed on the foot-wall, the bearings were made from two pieces of 1×3-in. steel, 6 in. long. A half cylinder was cut from a side of each piece and the two spaces together formed a bearing. Oil holes were provided, and in some cases holes were bored through the two pieces so that they could be screwed or spiked to the wall plates. The later practice was to forge the bearings from 1×3-in. steel, and to drill two holes at each end for $\frac{3}{4}$-in. lagscrews, by which the bearings were fastened to the timbers. These bearings were finally used on both foot and hanging wall. Similar idlers were so placed as to protect the dividers and end plates.

The difficulty of proper oiling presented the greatest obstacle to satisfactory results from this type of idler. As the clearance between the skip and the hanging-wall plates was sometimes less than an inch, there was not room for large oil or grease cups. In addition, the bearings were liable to get full of grit, especially when wet ore was being hoisted. Grease cups were generally unsatisfactory, although several kinds of grease were tried, and especial attention was paid to having that which was suited to the temperature of the shaft. In any event, it was necessary that the rolls be examined and the oil cups filled every two days, which meant the cessation of hoisting for two hours. The bearings wore

rapidly and the rollers tended to get out of line. The full skip weighed over 3 tons, and where the shaft flattened near the surface, the pressure against the idlers was heavy. It was only by distributing this weight over idlers placed but 5 ft. apart that anything approaching satisfactory service could be obtained at this point.

To obviate the necessity for so much attention, the idler shown in Fig. 282 was devised. The roller is extra-heavy 6-in. pipe, 3/8 in. thick, 20 in. long, in each end of which is pressed a cast-iron head, and through which passes a 1¼-in. steel shaft. This turns in a self-lubricating bearing carried by a bracket in a ball-and-socket shell, which prevents cramping. As a preliminary to adopting these bearings, two types of graphite and bronze self-lubricating bushings were tried side by side in the incline for three months. The one proving most satisfactory had cylindrical

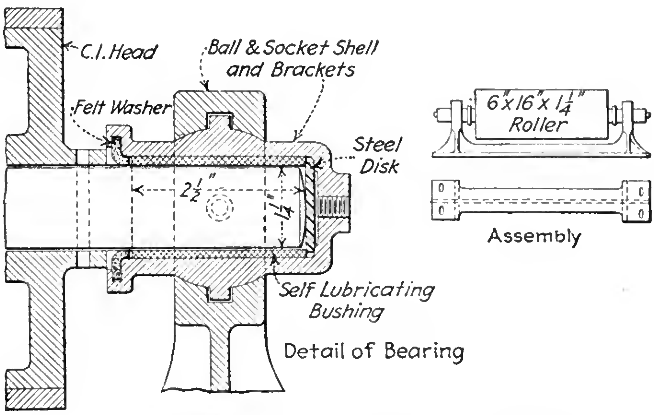


FIG. 282.—IMPROVED ROLLER AND ITS BEARING.

bodies of graphite ¼ in. in diameter set in the bronze, or "metalline," bushing at about ¾-in. centers. One end of the bearing is entirely closed, the end thrust being taken by a steel disk, which also serves for forcing out the bushing when it is worn. The other end of the bearing is protected from grit by a felt washer. This, however, also retains the fine particles of metal and graphite, and in time this gummy matter causes the bearings to bind. Occasional cleaning of the bushings with kerosene obviates this trouble. The cap is hinged at one side and fastened at the other with a hinged bolt so the roller and bushing can be easily removed. The bearings can be turned through 90°, and the roll turned end for end, permitting the advantage of full wear. The whole is carried in a casting which is fastened to the wall plates with ¾-in. lagscrews. These idlers have been in use nearly a year and are satisfactory. While with rapid and continuous hoisting the bearings become quite hot, they do not bind if they are cleaned occasionally.

For the lower part of the shaft, where the rope runs true and the inclination is 78°, the idlers are merely common sheave wheels, cast solid and keyed on a shaft of 1-in. cold-rolled steel. This has a total length of 13 in. These sheaves are 9 in. in diameter, with a 3-in. face, having a groove 1½ in. deep, 1 in. wide at the bottom The bearings are maple blocks, 4 × 4 × 6 in., bored to receive the shaft, and provided with an oil hole. These are fastened to the wall plates with six spikes.

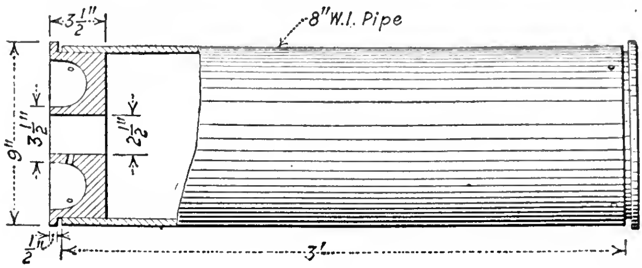


FIG. 283.—PIPE SECTION FITTED WITH CAST ENDS.

The wood-filled pipe idlers with forged bearings cost at Butte about $8 each, including bearings. The idler with the self-lubricating bearings costs $15, but the difference is quickly saved in decreased cost of attention. The solid cast sheaves weigh 26 lb. They cost, when fitted with a shaft, but excluding the maple bearings, $2.75 each.

Roller of Pipe.—A satisfactory horizontal or vertical roller may be made from a piece of discarded pipe by fitting the ends with castings, as shown in Fig. 283. These castings may be kept in stock for the several

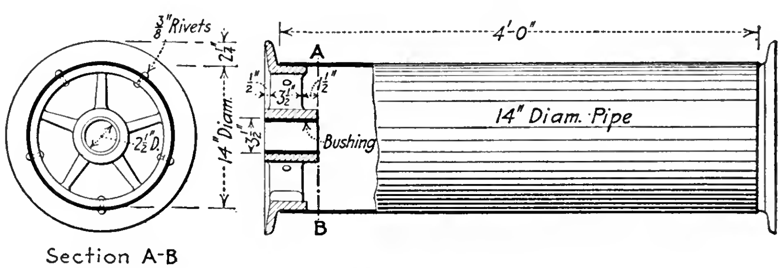


FIG. 284.—GUIDE ROLLER MADE OF PIPE AND OLD WHEELS.

diameters of pipe, and in case there is considerable lateral movement to the cable tending to make it leave the roller, the castings should be made with high flanges.

Roller of Pipe and Wheels.—Skip- and tram-car wheels discarded because of flange or tread wear may be utilized for rollers by driving the tread of the wheels into the ends of wrought-iron pipe of proper diameter and length. The pipe is then riveted to the tread of the wheels, as shown

in Fig. 284. This type of roller is particularly suited for rope carriers on inclined hoistways where the rope has some lateral travel, for knuckles and for side rollers to guide the rope around horizontal curves.

Rope Guide to Foot-wall Sheaves (By Clarence M. Haight).—A simple but effective method of guiding a hoisting rope to its sheaves on the foot-wall of an inclined shaft is shown by the accompanying drawing, Fig. 285, which explains itself. The brackets are used by the New Jersey Zinc Co. on its main shaft at Franklin Furnace, N. J. This shaft dips about 50°; the tracks are 6½ in. deep, and are supported on concrete piers. The guides are made of ⅞-in. round iron, supported by angles, as shown. The sheaves are of iron with a wearing surface of hard rubber.

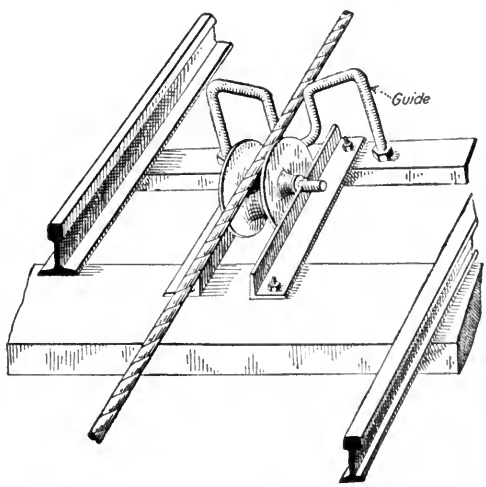


FIG. 285.—BRACKET FOR GUIDING HOISTING ROPE TO SHEAVE.

Substitute for Rollers in Incline (By H. H. Hodgkinson).—To pull rock or ore up grade in cars for 300 or 400 ft., a small drill-column hoist was used, having a rope speed of 80 ft. per minute with a pull of 500 lb.; it was found, however, that the hoist on being tested to the limit had a pull of over 1000 lb. at about half the speed. Because of the weight to be pulled and the small diameter of the hoist drum, a ¼-in. galvanized plow-steel rope was used. So small a rope had to be kept from dragging upon the ground or it would soon have worn out. Rollers were first tried for keeping the rope off the bottom, but proved unsuccessful, inasmuch as the weight of the rope was not enough to revolve them. Furthermore, although they were only about 12 ft. apart, the rope dragged on the ground between them when the car was lowered.

Therefore, as a substitute for the rollers, an old ½-in. hoisting rope was stretched from the point where the cars were loaded to the point

where they were dumped, firmly anchored at each end and tightened by means of two turnbuckles, Fig. 286. This rope was 8 ft. above the center of the track; on it double-wheel trolleys were hung at 20-ft. intervals; the upper wheels traveled on the ½-in. rope, while the lower wheels carried the ¼-in. hoisting rope. As the car was hauled up, these trolleys all advanced along the ½-in. rope until the car reached the dumping point. The trolley wheels were ⅝ × 3-in. pulley-block wheels with ½-in. bolts as shafts; the yoke was made of ⅜ × 2-in. iron. The grade, however, was not great enough to cause the trolleys to travel

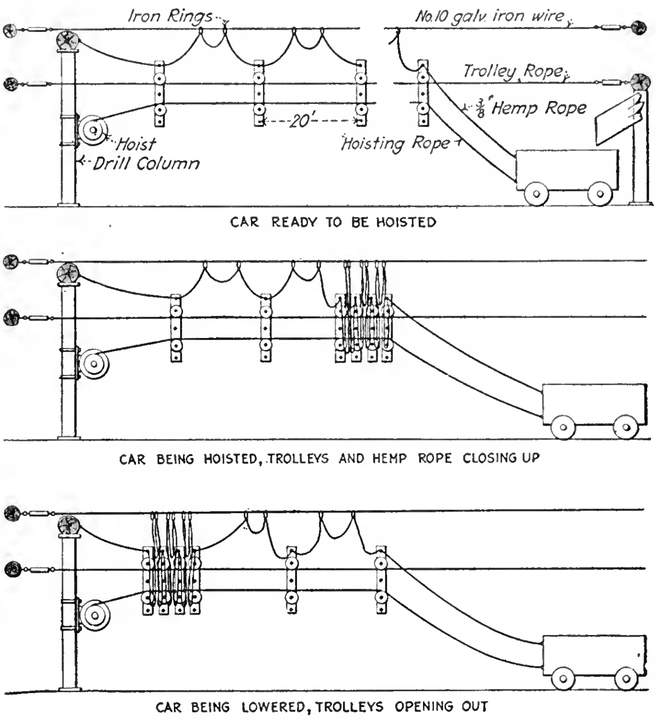


FIG. 286.—DIAGRAMMATIC REPRESENTATION OF INCLINED HAULAGE INSTALLATION.

down the incline. In order to make them take their respective positions again, a No. 10 galvanized iron wire was suspended at the same elevation and parallel to the ½-in. rope, but 3 ft. to one side, and was also tightened. In order to avoid confusion, it is shown in the illustration as placed above the ½-in. rope. On this wire were placed two small iron rings between each pair of trolleys. A ⅜-in. hemp rope was then used to connect trolleys and rings in series as shown. The last trolley, that nearest the car, was connected to the car by means of the same rope. The descending car by means of the hemp rope pulled the trolleys back down the ½-in.

rope and spaced them properly. The rings on the small wire looped up the hemp rope and took up its slack so that it did not become tangled, or drag on the ground.

Sheave-wheel Lining (By G. L. Sheldon).—The rubber lining used to lighten the wear on sheave wheels and to keep wire cable from slipping, frequently cuts out when the wheels are run at short angles. The following method of relining the wheel will be found satisfactory after all of the old pieces of rubber have been removed. Unwind an old piece of large-size manila rope and wind into the sheave wheel a single strand of the rope, applying a steady tension. As each layer of rope is run on the wheel saturated it with pine tar. Continue the operation until the lining reaches the desired thickness. A liberal allowance of tar should be applied to the last layer of rope and the wheel should be allowed to stand from 12 to 18 hr. before using. This lining serves just as well as rubber and is far cheaper. Coal-tar or melted pine-pitch gum may be used in place of the pine tar.

ROPE

Reversing Rope on Single-drum Hoist (By R. S. Schultz, Jr.)—It is well known that the life of a hoisting rope can be considerably lengthened by reversing it at the proper time. This is due to the fact that the weight of the rope itself brings a heavier load on the drum end of the rope and causes that end to fatigue more rapidly. With a two-drum hoist, reversing the ropes is a simple process, but with a single drum, and especially with a long rope of large diameter, the change becomes more of a problem. By using two old cable-reels, reversal can be made by winding the rope from the drum on to one of the reels, rewinding to the second reel and then rewinding on the drum, but this is a slow, tedious operation and requires considerable preparation.

The following method is simple, comparatively rapid, and requires little or no preparation other than clearing a small space in front of the engine house. The skip or cage is hoisted to the collar of the shaft, securely fastened, and the rope detached. The end, just above the socket, is tightly wound with wire to prevent raveling; the socket is cut off and the rope wound over the sheaves on the drum until the end is at the cleared place in front of the engine house. The hoist is then reversed, and the rope coiled on the ground in a large figure eight. Then the coils are taken a few at a time and thrown over until the whole rope is reversed. The former skip-end is then refastened to the drum, the rope rewound on the drum, the end pulled over the sheave, the socket rebabbitted and the rope made fast to the skip or cage. Special care must be taken to bend the rope naturally in coiling and uncoiling, otherwise a serious kink may result.

In one case, using this method, seven men reversed 3000 ft. of 1⅛-in. plow-steel rope in about four hours, including rebabbitting the socket on the skip-end. Considerably better time could have been made, had speed been necessary.

Reversing Rope with Single Coil (Joseph Hocking).—There is a better method than that of Mr. Schultz for reversing a rope on a single-drum hoist, provided that the rope is no longer than 2000 to 4000 ft., and that there is a space about 50 ft. square available outside of the engine house or shaft house. The method consists of starting a coil about 10 or 15 ft. in diameter and continuing coiling, working toward the outside until the rope is all off the drum; it is easy to lay three coils to a foot. With the rope off the drum, all that is necessary is to cut the socket end from the rope, fasten the socket to the other end, which will be at the outside of the coil, fasten the inside end to the drum and wind it up.

Reversing Rope, Using Power of the Hoist (By C. M. Rasmussen).—It is not necessary to do by hand all the hard and dirty work of reversing a rope, when the engine can do it. If the rope has to be rewound on account of slackness on drum, or the wearing parts in the rope have to be shifted, or the rope has to be turned end for end, then the following is the best, cleanest and quickest way a rope can be reversed, requiring less than two hours for one from 4000 to 5000 ft. long:

Block the skip or cage at the surface; unfasten the end of the rope and coil up enough rope to go over the headgear and be fastened to the engine drum later. Then for a 1¼-in. diameter rope get a 20-in. single-sheave wheel, unturnable, and put around the wheel the bight of the slack rope coiled up. Fasten the block, unturnable, to the bridle of the skip. Fasten the free end of the rope above the skin with strong chains or clamps. Then let the engine pull the rope tight, take the blocking away from under the skip and let the skip run down the shaft with double rope. When all the rope is off the engine drum, fasten this end of the rope in the headgear also with chain or clamps. Then unfasten the former end and pull the one end down and the other up over the headgear and fasten to the engine. When ready pull the rope tight, take the chain off the end of the rope first made fast, signal to the skipman below to fill the skip at any handy ore box with rock, let any slack rope already on the drum off again by lowering the skip and then pull the loaded skip to the surface. Block the skip and take the 20-in. sheave wheel off the skip, and unchain the new end of the rope and fasten it to the skip. Take the blocking from under the skip, and the job is finished. All this work seems complicated on paper, but when once tried will be found easy and can be done in much less time than coiling a mile of rope on the ground. If the rope requires lubricating, that can also be done at the same time.

Put the grease box across the shaft, and pass the one rope going down through the hot grease.

Lubricating Box for Horizontal Hoisting Rope (By R. B. Wallace).—An apparatus applicable to lubricating a hoisting rope which is horizontal or nearly so in some portion of its course, is illustrated in Fig.

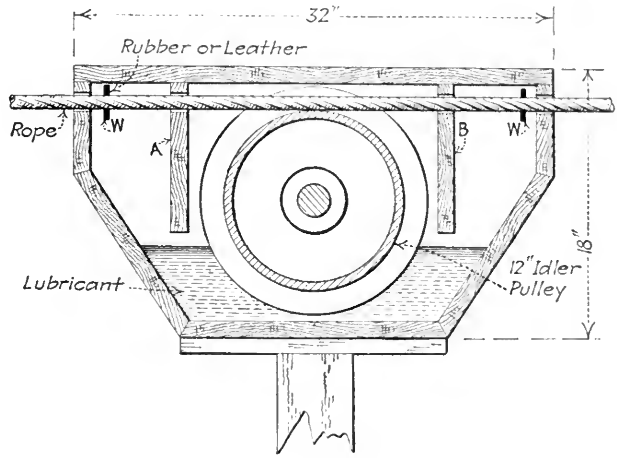


FIG. 287.—BOX FOR LUBRICATING FLAT-LYING HOISTING ROPE.

287 in longitudinal section. It consists of a wooden or metal box which has a removable cover and is bored so that the rope can pass through. It is $7\frac{1}{2} \times 18 \times 32$ in. as shown. The lubricant is contained in the bottom of the box and is picked up on a wheel and carried to the rope, the wheel being actuated by the rope running over its top. The wheel

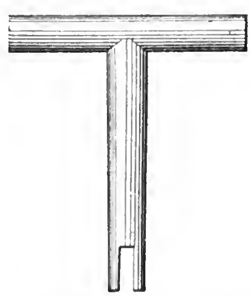


FIG. 288.—WRENCH FOR ROPE WIRES.

is a cast-iron idler sheave running free on a 2-in. shaft which is held in holes in the box sides by cotter pins. There are two rubber or leather washers *W*, partially free, which serve to clean the rope of excess oil.

Tool for Bending Rope Wires in Socket Connection (By Joseph Goldsworthy).—The method of fastening a hoisting rope into a conical

socket is well known. As usually practised it involves bending each wire in the rope end inward to form a hook. For forming the hooks, a piece of round steel, made up in the shape of a T with a slot in the end, as shown in Fig. 288, will be found more convenient to use than pliers.

Single-screw Wire-rope Clip (By A. Livingstone Oke).—Fig. 289 shows a new form of clip for fastening wire ropes together. It consists of a steel link, made flat as shown in the section. The sides of this link fit in two grooves or channels cut in the inside of a nut. The inside dimension of the link is just sufficient to clear the tops of the threads of a plug which works in the nut. The outer end of this plug is square so that it can be turned by a spanner. Between the plug and the two ropes is a piece of iron grooved slightly on one face, to fit neatly on the rope which comes next to it. The ends of the link have wings on them. If it is impossible to pass the ropes through the link when the nut is on,

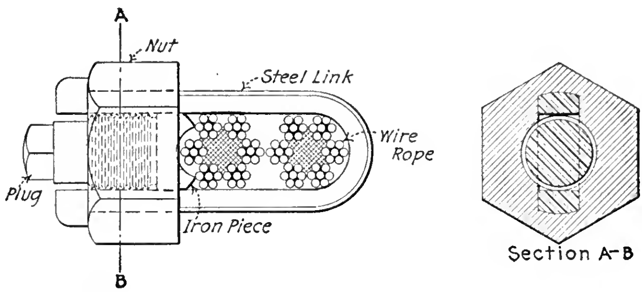


FIG. 289.—CROSS-SECTIONS THROUGH ROPES AND CLIP AND THROUGH NUT.

then a special link with long ends instead of the wings can be employed, the ends to be hammered back when in position on the ropes.

TRANSPORTING

Drag Scraper for Handling Dump (By Frederick W. Foote).—The Portland mill, at Victor, Colo., was erected to treat the low-grade dump material from the mine. The dump contained about 2,500,000 tons with an average value of about \$3 per ton. It was found advisable to take the supply from the top of the dump. A cheap and efficient means was devised and put into operation whereby the material was elevated to the top of the dump. It consisted of a drag scraper, Fig. 290, connected to a 1⅛-in. endless cable which ran through a pulley fastened at the bottom of the dump and around the hoist drum at the top. An electric hoist of 112 hp. was used and the cost was about 1 ct. per ton of material raised. The scraper required about two days for its construction. The body was of ½-in. boiler plate and the teeth of cast

iron. The life of such a scraper depends on the character of the material handled and the speed of handling. The one described handled loose rock, and, in almost continuous operation, lasted from two to three weeks.

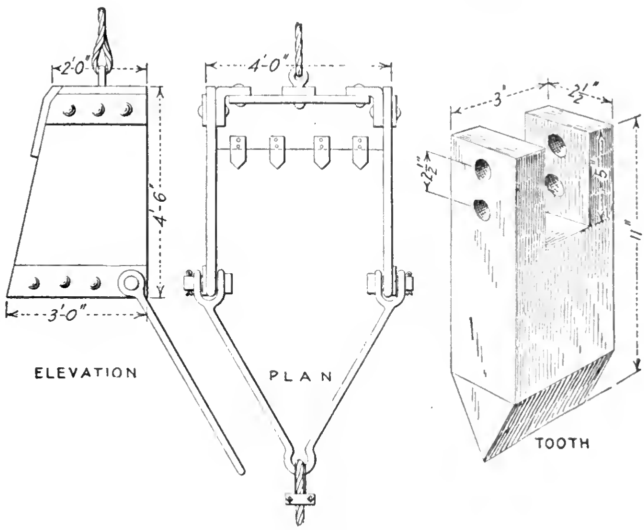


FIG. 290.—CONSTRUCTION OF SCRAPER AND DETAIL OF TOOTH.

Single-track Cableway (By E. Praetorius).—A cableway of unusual design, illustrated in Fig. 291, was installed at the Rosas mine in Sardinia. Its novel feature consists in the fact that the two buckets travel on the

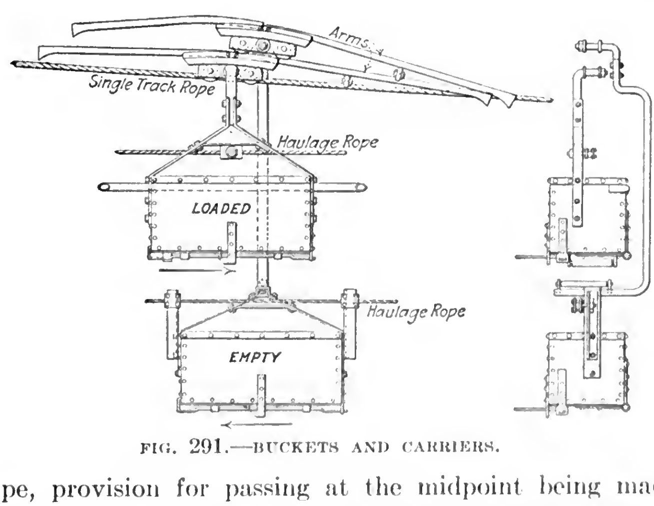


FIG. 291.—BUCKETS AND CARRIERS.

same rope, provision for passing at the midpoint being made in an ingenious manner. To the carrier of each bucket are attached two arms extending parallel to the track rope, and above it. These arms are pivoted

where attached to the carrier and the one on the ascending side is kept elevated above the rope by means of a flat steel spring inserted below it. The tops of these arms form tracks which take the wheels of the other carrier just as the rope does. When the buckets meet, the carrier wheels of the ascending bucket mount the carrier arms of the descending bucket and ride over them. The spring in the upper arm yields under the weight of the ascending bucket, so as to permit the arm to come down on the rope and deliver the carrier wheels to the rope again. The ends of the arms are made fantailed so as to guide them to the rope. One bucket is

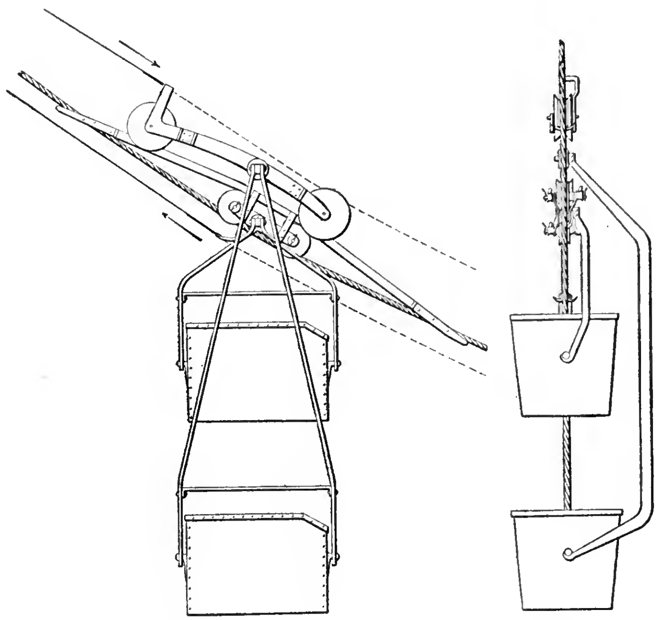


FIG. 292.—DEVICE PERMITTING TWO BUCKETS TO PASS ON SINGLE ROPE.

suspended from the carrier by a much longer arm than the other, this arm having an offset portion to permit the passage of the other bucket.

The length of the cableway is 1115 ft. and it overcomes a difference in elevation of 223 ft. The track rope is $\frac{3}{4}$ in. in diameter and the haulage rope is $\frac{5}{16}$ in. The weight of each bucket and carrier is 220 lb. and it carries a load of 660 lb. The speed of the buckets is such that the 1116 ft. is covered in 65 to 70 sec.

Single-track Cableway (Herbert K. Scott).—Fig. 292 illustrates a single-track cableway similar to that described by E. Praetorius and no additional description is necessary more than the noting of the fact that in this case the carriers for the two buckets differ in design and the one

shown on top always rides over the other whether it is descending or ascending.

ACCESSORIES

Loading Derrick at Shaft Collar (By Clarence M. Haight).—At the Palmer shaft of the New Jersey Zinc Co.'s mine at Franklin Furnace,

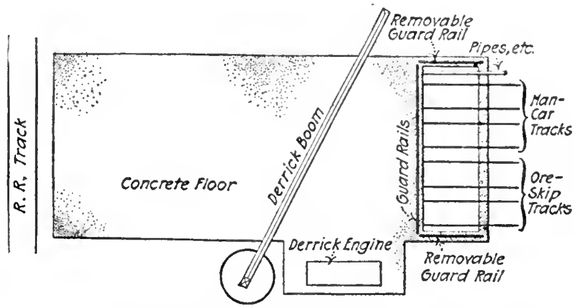


FIG. 293.—RELATION BETWEEN DERRICK, SHAFT, TRACKS, ETC.

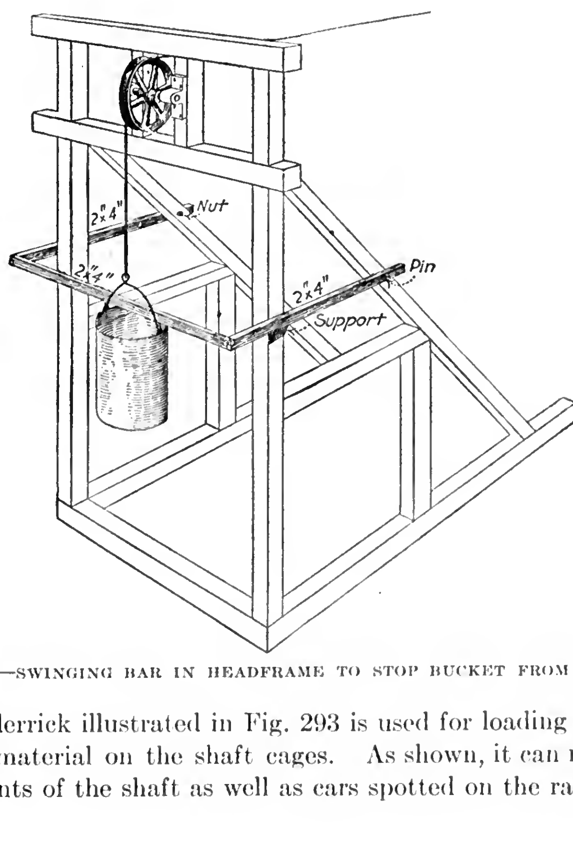


FIG. 294.—SWINGING BAR IN HEADFRAME TO STOP BUCKET FROM SWINGING.

N. J., the derrick illustrated in Fig. 293 is used for loading timber, rails and other material on the shaft cages. As shown, it can reach all the compartments of the shaft as well as cars spotted on the railroad track.

The mast is made of 14 × 14-in. timber and is supported by six guy lines; the boom is made of 12 × 12-in. timber. When not in use the boom is supported on a post. The power is furnished by a two-cylinder Lambert engine, operating on compressed air.

Device to Stop Whirling Bucket.—A simple contrivance to stop the whirling of a bucket, and thereby save time and trouble, where hoisting is being done without a crosshead, is shown in Fig. 294. It consists of two pieces of 2 × 4-in. lumber pivoted on the backstay of the headframe by pins and joined together, in front of the headframe, by a third 2 × 4-in. piece. The stop thus is free to swing upward, and is prevented from falling by two supports on the headframe posts. The outer face of the crosspiece is almost flush with the hoisting cable, and the piece is level with the top of the bucket when it is at an elevation convenient for dumping.

When hoisting, the bail of the bucket may strike the stop and lift it temporarily, but when the bucket makes a quarter turn the stop will fall alongside the bail and prevent any further whirling.

IX

SHAFT CONVEYANCES

Cages—Skips—Chairs and Dogs—Skip Dumps—Transfers—Buckets—Bucket Dumps

CAGES

Drop-bottom Cage.—In Fig. 295 are shown the details of the drop-bottom cage used at the mines of the St. Louis Smelting & Refining Co., in southeastern Missouri. The notable feature of the cage is that the car is locked in position on the deck by dropping the central part of the deck track. In landing, this track is raised up level with the outer rails before the cross-braces of the deck of the cage come into contact with the landing chairs. The outer fixed rails are given a slight slant toward the center of the cage so that, if the car has not been put on properly, the jar of lifting the cage off the chairs will cause it to run to the center and drop down into the recess. Only its weight holds this movable portion of the deck in place and in case of repairs it can be easily lifted out. Pans marked P are riveted to the bottom of the deck so as to come up alongside the rails and keep anyone from getting his toe under the rails of the drop portion. In this way possibility of injury from that source during the hoisting of a deck load of men is prevented. The possibility of a car getting loose in the shaft and causing a wreck is almost completely eliminated with this device. A disadvantage is that the use of chairs in the shaft is entailed.

The cage is equipped with the ordinary type of cam safety catches operated by coil springs and chains from a collar on the drawbar. The side plate is punched to receive the cam shafts and extends down the side of the cage to carry the angle-iron guide shoes S. These shoe angles extend clear down under the deck of the cage so as to tie the deck securely to the sides. To the bottom, where the angles are bent around, reinforcing plates G are riveted. The deck of the cage is made up of the outer angle cross-braces M, the inner braces N, and the shoe angles S. The drop bottom is made up of the cross-braces K and L, the angle irons I, the longitudinal angles H bolted together back to back and the rails R_1. In landing the chair shoes catch the cross-braces L and raise the drop bottom even with the rest of the deck.

Cage with Munzner Safety Catches.—Cages fitted with the Munzner type of safety catch have been in use for 15 years in the shafts of the Doe

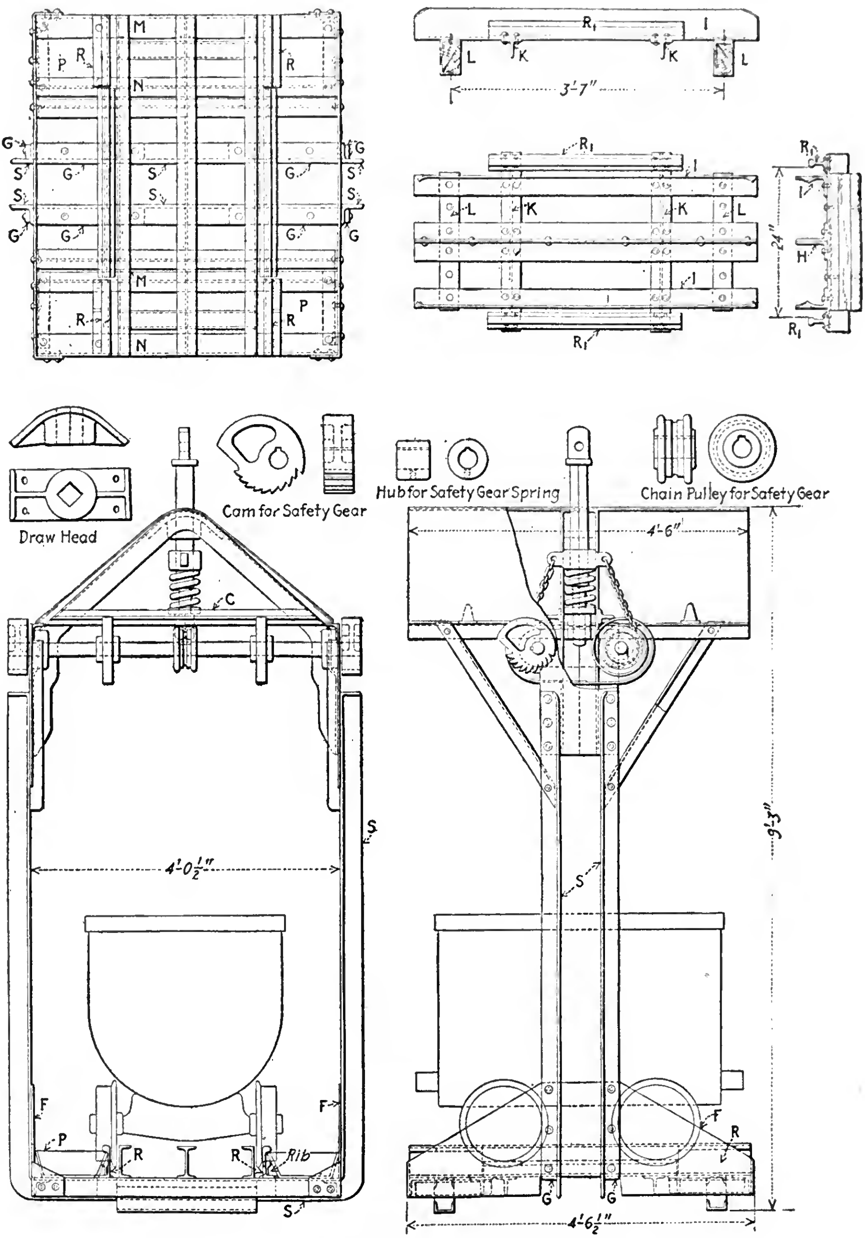


FIG. 295.—ST. LOUIS SMELTING & REFINING DROP-BOTTOM CAGE.

Run Lead Co., of southeastern Missouri. This safety catch was designed by F. A. Munzner, in Germany, about 1893, and is largely used in Saxony. Its action consists of thrusting pointed knife edges into the guides by a toggle action. The shape of the dogs used in the Doe Run mines was worked out by Karl Kley, in Saxony, and was adopted, as being the most satisfactory, by O. M. Bilharz, general manager of the Doe Run company. The dogs offer the advantage as against the ordinary type of toothed-cam safety dog, of stopping the cage with a slower, braking action instead of with a sudden grip and they more perfectly fulfill the requirements of a good safety catch, namely, that it be positive and reliable, quick to come into action, but slow to complete its action, and capable of acting on guides of varying thickness. It is urged against the cam dog that it acts so quickly as to tend to injure the men riding on the cage and to tear out the guides. It also is likely to fill with wood and possibly thus become inoperative. The use of the double knife edge minimizes the danger of splitting the guides and while it cuts and damages them somewhat more than a single blade, this is a point of minor importance.

On one occasion at the Johannes shaft in Freiberg, the Munzner attachment stopped a cage so gradually after the cable broke, that the men riding did not realize that it had not been stopped by the engineer. In demonstrating the reliability of the device to the Doe Run miners, when it was first proposed to install it, Mr. Bilharz and his master mechanic mounted the cage 40 ft. from the ground in an experimental tower, cut themselves loose and were stopped immediately. In the 15 years' use of the Munzner catch in the Doe Run mines, the only instance in which it failed to work was on one occasion when the rope broke with the cage 20 ft. from the shaft bottom and the catches were unable to take care of the extra weight of cable and stop the cage in that distance. Usually the cage stops in 3 or 4 ft. In a German test, a cage weighing 3440 lb. was allowed to fall about 1.5 ft., when the safety dogs came into action and sank 1¼ in. into the guides, stopping the cage after a groove about a foot long had been cut.

On the Doe Run cages, the spring is placed above the actuating frame instead of below, an improvement on the original design. The details of the device are shown in Fig. 296. The dogs *D* are carried loose on the ends of the shafts *B*. The shafts are pinned to the crosshead *A* and cross-braces *C*, making a rigid frame, which moves with the drawbar *F* to which *A* is keyed. The dogs are held on the shafts by collars and rest on and move over the plates *E*. The spring *V* is held under the strap *H*. When the pull comes on the drawbar, it moves up, and the collar *I*, keyed to it, compresses the spring *V* until *A* strikes the plate *P* fastened to the cage frame and thus gives a positive stop to upward motion of

the drawbar and the frame. This motion tends to pull the points of the knives D away from the guides. If the rope breaks or the tension is otherwise released, the frame is thrust down, relative to the cage, by the spring V and the dogs revolving on B and lying on E are forced with a toggle motion to engage the guides. It is necessary to keep the knives always sharp.

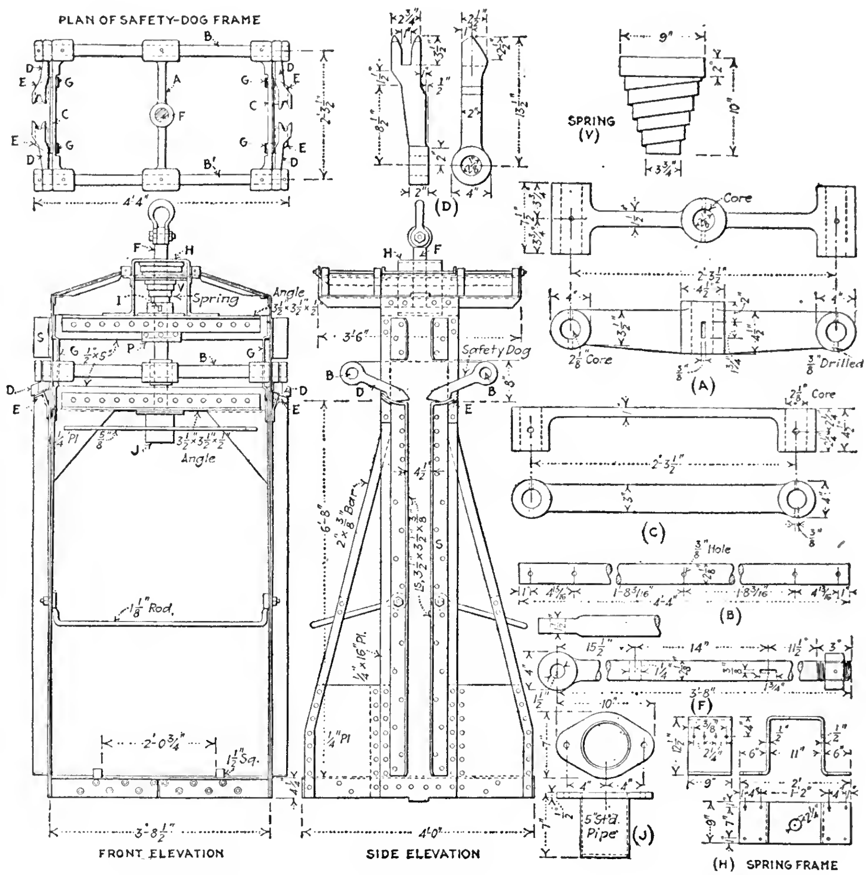


FIG. 296.—THE MUNZNER SAFETY DEVICE AS APPLIED TO DOE RUN CAGES.

The bottom of the drawbar is protected by a cylindrical hood J to prevent pinching the hands of the men riding. The motion of the frame and the dogs is controlled by the vertical guides G. The original design had stops to limit the inward motion of the dogs. These are here omitted. The guides in this case are 4 × 6 in. The spring is a volute, 9 × 9½ × 2 × ⅜ in., made by the A. French Spring Co., of Pittsburgh, and has proved satisfactory. The spring should not take the whole load

of the cage before the frame catches P, or the device will be too sensitive. A sharp fluctuation in speed might tend to make the dogs catch and the jumping of the cage after stopping would bring them into action. Probably the adjustment should be such that the spring will take about 80 per cent. of the weight of the empty cage before the positive pull comes

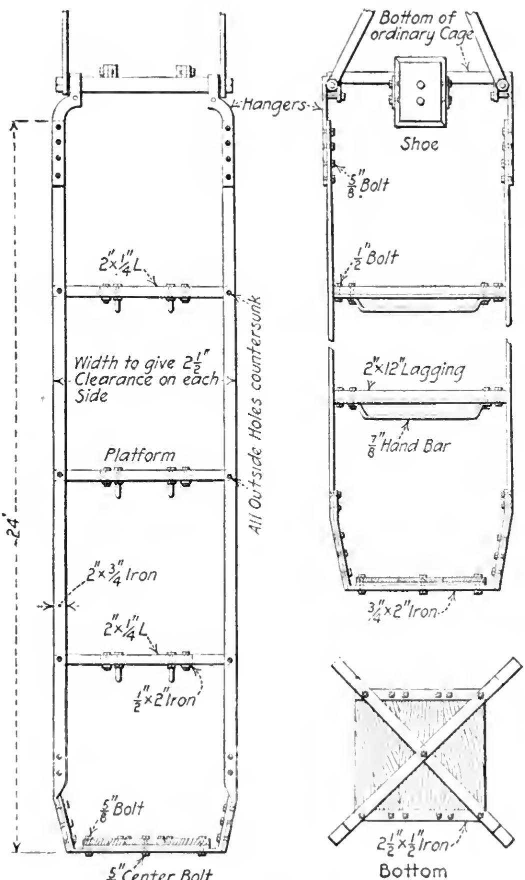


FIG. 297.—SKELETON CAGE, 24 FT. HIGH.

on the frame. The best springs will deteriorate and should be frequently tested to determine their strength quantitatively.

The weight of the cage illustrated is 2400 lb. and it cost, as shown, about $425, erected in the company's shops. One difficulty with the design is the obstruction of the top of the cage, which interferes with loading timbers, pipe and rails. The knife-edge principle, however, should be applicable to a cam dog with satisfactory results; possibly in

such case, the edges should be toothed slightly at the points where they begin to grip.

Four-deck Shaft-repair Cage (By Albert B. Pedersen).—Fig. 297 shows a 24-ft. cage with four decks used by the Chief Consolidated Mining Co. at Eureka, Utah, for changing guides from $3\frac{1}{2} \times 3\frac{1}{2}$ in. to $3\frac{1}{2} \times 7$ in. After this work was done shoes were put on the bottom and the cage was used for regular shaft inspection and repair work. The long cage was swung from the bottom of the ordinary cage by means of the hangers at the top. The cage corners were made of $2 \times \frac{3}{4}$-in. strap iron; $2 \times \frac{1}{4}$-in. angles attached to the corner irons supported wooden platforms; to the bottoms of these were bolted hand bars of $\frac{7}{8}$-in. round

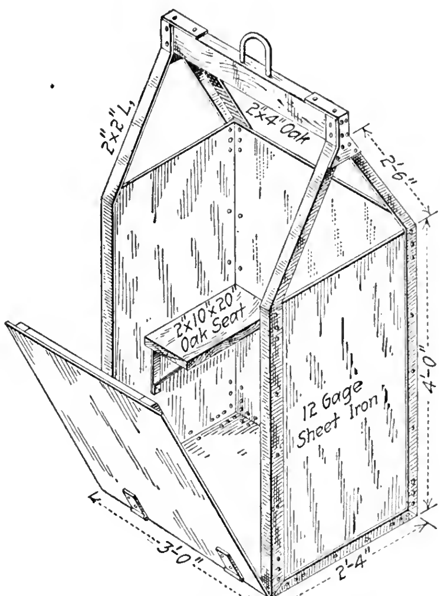


FIG. 298.—CAGE FOR HOISTING INJURED MINERS THROUGH TIMBER SHAFT.

iron. The bottom was made tapered as shown. Using this cage with one man on each deck, about 200 ft. of guides was changed on an average in eight hours.

Ambulance Cage (By L. D. Davenport).—If a miner working on one of the sublevels in the Chisholm district of the Mesabi range is injured, it is usually easier to hoist him up one of the timber shafts than to take him to the bottom level and hoist him up in the skip. At one time it was customary to use the powder cage for this purpose but this was so small and inconvenient that a special type of apparatus was designed. The cage shown in Fig. 298 is of sufficient height so that an injured man can stand or be supported in an upright position if desired. The large door

in the front, which is reinforced with oak cleats, can be swung down to the ground, making it easy to place an injured man in the cage. There is a seat on one side and the cage is made large enough to accommodate two men in case it is necessary to have the injured man accompanied by someone. The cage is light and strong and was designed and built under the direction of the master carpenter at the Monroe mine.

Latch for Holding Car on Cage.—Fig. 299 shows a device for holding timber or other trucks on the cage, employed at the Kennedy mine on the North Cuyuna range. It consists of a horizontal bar of round iron held loosely by staples to the wooden floor of the cage, so as to revolve freely but not to slide. Near one end a 1¼-in. round bar is welded on to project at right angles and its end is split to form a fork 5 in. across the

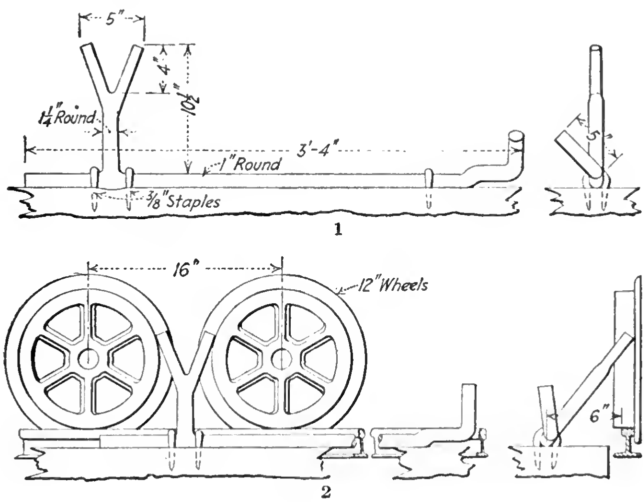


FIG. 299.—AN EFFICIENT CAR LATCH.

top. The other end of the horizontal bar is made eccentric for a few inches and then turned at right angles. The latch is placed so that the fork will be at about the center of the cage and the planes of the fork and of the bent end are so related that the operator can slip his foot under the latter when the fork is thrown back out of the way. The construction and dimensions are shown in 1, Fig. 299, which, however, does not represent the latch in a normal position. In 2 the latch is shown engaging the wheels of a truck. The truck is run on the cage until the space between the wheels is about opposite the fork, when the operator throws the latch over and the fork falls on the treads of the wheels and against the flanges, at a point somewhat above their centers. The truck is thus held against movement in either direction. The device is in use on a slow-moving cage in a shallow mine. It would seem that on a fast-moving

cage, one, for instance, actuated by a first-motion hoist, there would be danger that the fork might bound out as the result of a jolt and thus release the truck.

Releasing Hook for Cage Testing.—Usually, when testing the safety catches on skips or cages, the hoisting cable is attached to the cage by a hemp rope which is cut when the cage is in the desired position. The inspection department of the Cleveland-Cliffs Iron Co., in the Lake Superior region, tests all safety dogs once each month. A releasing hook is used instead of the hemp rope and has proved more satisfactory. Fig. 300 shows the details of the hook, which is strong enough for a 5-ton skip

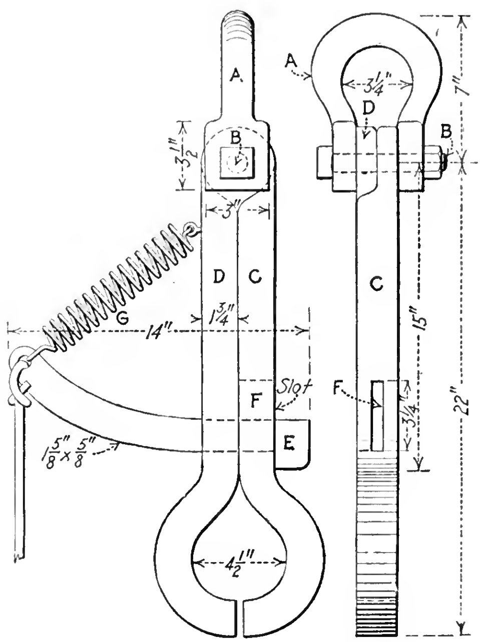


FIG. 300.—CLEVELAND-CLIFFS RELEASING HOOK.

of the Kimberley type. The trigger E is pinned to the arm D and operates in the slot F cut in the arm C. The trigger catches on the lower back edge of this slot. The pin B of the clevis A, which fastens the cage to the shackle of the hoisting cable, is removed and the clevis attached to the releasing hook. The hook is now linked into the drawbar of the skip or cage whose safety dogs are to be tested. When the cage has reached the proper position, the rope attached to the trigger is pulled, allowing the arms D and C to separate and the cage to fall. At the Cliff shaft of the Cleveland-Cliffs company, the safety-dog testing is done at the bottom of the shaft where actual working conditions as to slippery guides

are more nearly duplicated than at the surface. This practice is open to objection, however, if the bottom guides are not worn so much as those higher up. The danger due to the dogs cutting up the guides is better tolerated at the bottom of the shaft than at the surface. Should the dogs of a heavy skip refuse to work and allow it to fall on the stage at the collar of the shaft, damage might be done to the shaft timbers; this danger is obviated by the use of the bottom of the shaft for testing purposes.

Combination Cage and Skip.—Fig. 301 shows the details of the cage and skip used at the No. 1 shaft of the Federal Lead Co., in southeastern Missouri. There is a top deck which can accommodate a car and on which men can be handled, and under it swung from the same frame is a lower deck which carries the self-dumping skip. This skip holds 5.4 tons of ore.

There are several interesting features in the design. To the channels that form the main members of the cage and skip frames, four shoe channels are riveted, two on the frame that carries the skip, and two on the frame that carries the deck for the men and car. The two frames are bolted together by means of a splice plate. The main channels are bent over at the top and spliced together by a strap *A*2, through which passes the drawbar *A*1. Under the two runner channels and bolted to them is the drawhead casting *B*1. This is contracted at the center so as to be straddled by the clevis-shaped drawbar *A*1, which also straddles the shock spring and shock-spring base *B*2. The spring, of course, foots against the drawhead casting. This whole mechanism is inclosed by cover plates fastened to the channels. One plate is riveted while the other is fastened by bolts so that it can be easily removed. On the draw pin *A*3 is a sleeve over which fits the clevis by which the cage-and-skip frame is fastened to the rope shackle.

The chains leading down to the safety-dog levers fasten to eye-bolts, through which the pin *A*3 passes. The safety dogs, which are toothed castings, are held together by the spring *S*; this connects with the two lever arms *B*, which in turn are keyed to the dog shaft. The safety dogs are held away from the guides by the pull on the drawbar, which is transmitted to them by the chains connecting with the levers *A*, which are also keyed to the dog shafts. The rest of the cage follows the ordinary lines of construction.

On the lower platform of the cage, it will be noticed that there are two pivot shafts for fastening the skip body to the frame. This is to allow the skip to dump in either direction. Formerly it discharged into a pocket on the opposite side of the shaft from that now in use. A peculiarity of the skip is the use of the horns *H* to right it after dumping. The tripping is accomplished by the skip rollers following out on a

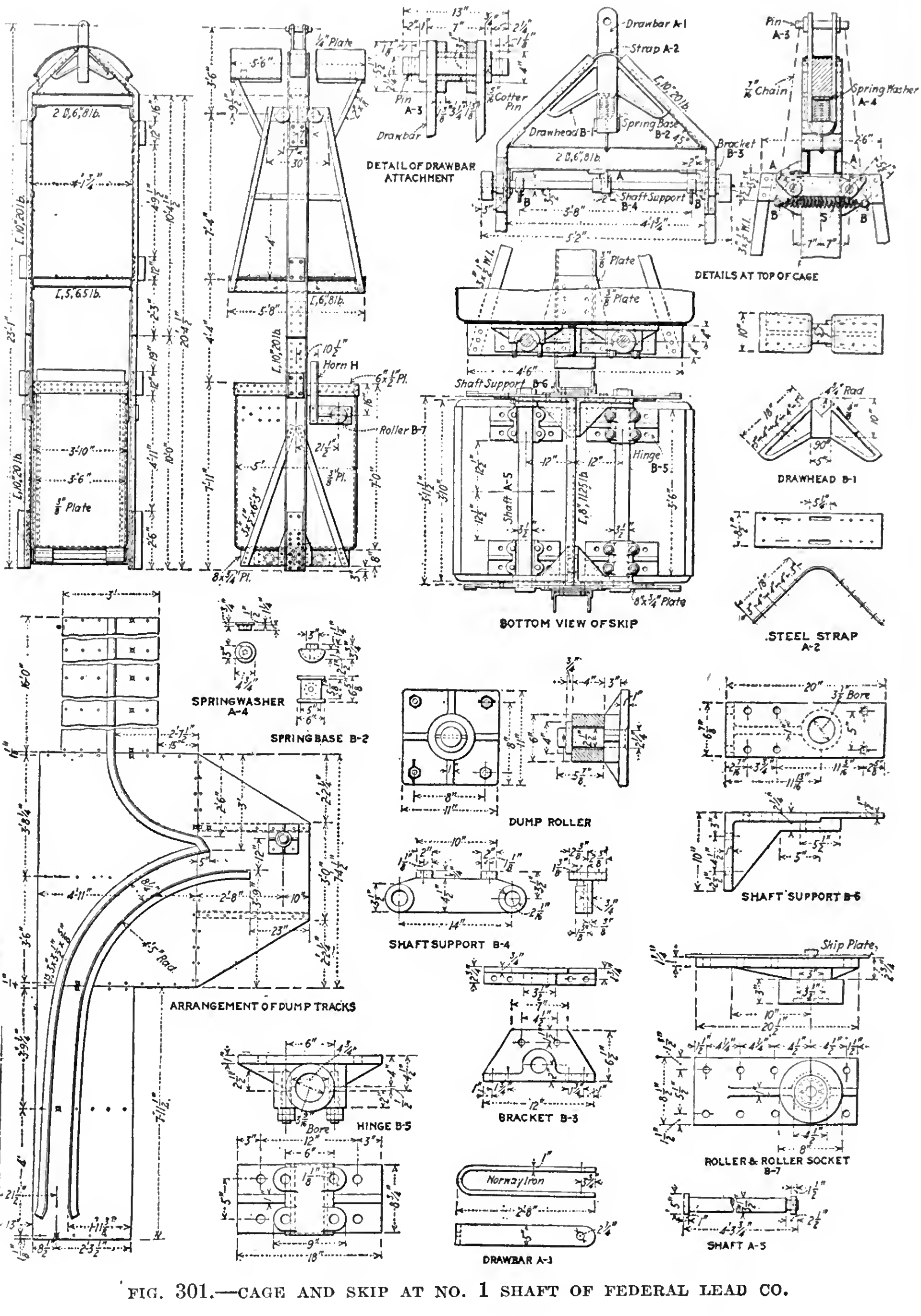


FIG. 301.—CAGE AND SKIP AT NO. 1 SHAFT OF FEDERAL LEAD CO.

double runway, as can be seen in the drawing of the dump, the upper runway, however, having only the bottom track. On coming down after dumping, the skip rollers run forward, following the guide runners until at the point of the dump tracks, the horns engage with the nose rollers of the dump, and the body of the skip is given a push upward, bringing it back into an upright position on the skip frame. The other features of the skip do not need mentioning, as they follow the ordinary lines.

SKIPS

Light Skip for 65° Incline.—Fig. 302 represents a $\frac{3}{4}$-ton skip designed for a Mexican mine. The rounded bottom is more common on vertical

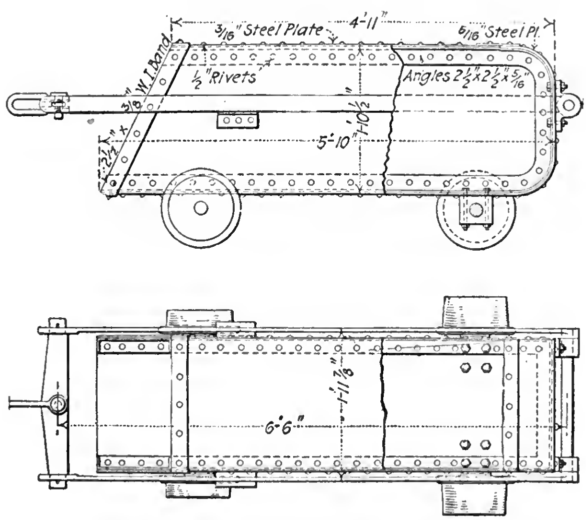


FIG. 302.—SMALL CAPACITY INCLINED SKIP.

than inclined skips. The extremely small cross-section, 1 ft. 10 in. by 1 ft. 11⅞ in., is noteworthy. The gage of track is 2 ft., the wheels are keyed to the 2-in. axles which revolve in hardwood boxes.

Rear-dumping Skip-car (By L. O. Kellogg).—At the Sterling iron mine, in Orange County, N. Y., the flat-dipping shaft and the limited room between the collar and the hoist make the employment of a special type of skip necessary. This is shown in Fig. 303. It is really a car with an automatic door on the bottom end. The door is shown closed; opening is effected by tripping, two small wheels coming into contact with a railing erected beside the track at the dumping point. The smaller wheel first comes into contact and unlocks the toggle joint. Then the

larger wheel forces the lower half of the door to slide up outside the fixed upper half. As the car descends again, the door is allowed to drop and is locked with the toggle mechanism. There is a double track with a tripping railing on each side and the tripping device on the cars is put on both sides, so that the cars can be used on either track. The car body is built with its bottom higher toward the upper end. By this means sufficient slope is given to the bottom to discharge all the contents.

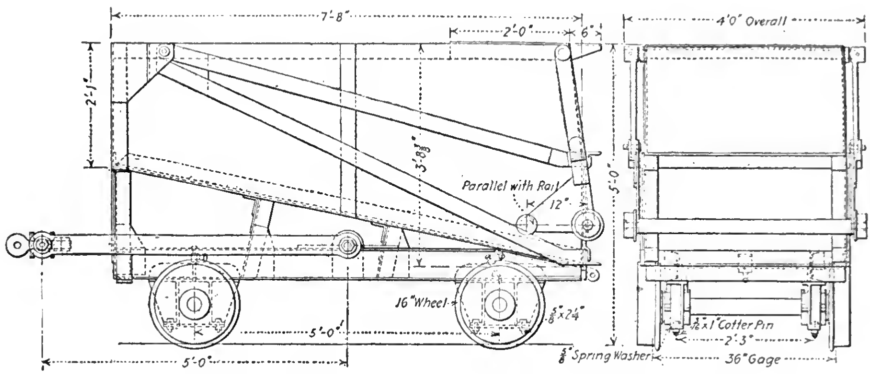


FIG. 303.—SKIP CAR FOR INCLINE, WITH REAR DISCHARGE.

The car, as used, hoists 4 tons of magnetite. It was built by the Kilbourne & Jacobs Manufacturing Co., of Columbus, Ohio, to fit the peculiar conditions existing, and is a unique type.

Skip-bail Lock and Release (By S. S. Jones).—The device illustrated in Figs. 304 and 305 is one designed to prevent overturning of the skip in the shaft. It was developed by W. H. Flannigan, foreman of the Tom Reed mine in Arizona. Two notched bars or latches, *A*, are normally caught over a square lug on each side of the bail and are rigidly connected to a square shaft *B* under the skip, turned for two bearings. A lug *C* projects from the middle of this shaft and a trigger *D* forked at the top is pinned to this. This trigger is of the shape shown. When the skip arrives at a point near the dump, the trigger is caught by a 2-in. plank covered with sheet iron and spiked between the rails. This forces the trigger to revolve until the upper part presses on the shaft, when it becomes in effect rigidly connected with it and the shaft is revolved sufficiently to release the latches *A* from the bail lugs. The trigger remains in contact with the plank after the skip starts on its dump. On the return, it again comes into contact with the plank, but is free to revolve in the opposite direction. The latches are thus allowed to fall by gravity on the lugs. They are tapered at the top so that the bail can

slip into place. This return of the latches might be made more positive by the use of a spring, but, as a matter of fact, this has not been found necessary. The mine shaft in which this is installed has a dip of 73°. The dimensions of the ½-ton skip and the parts of the device are not

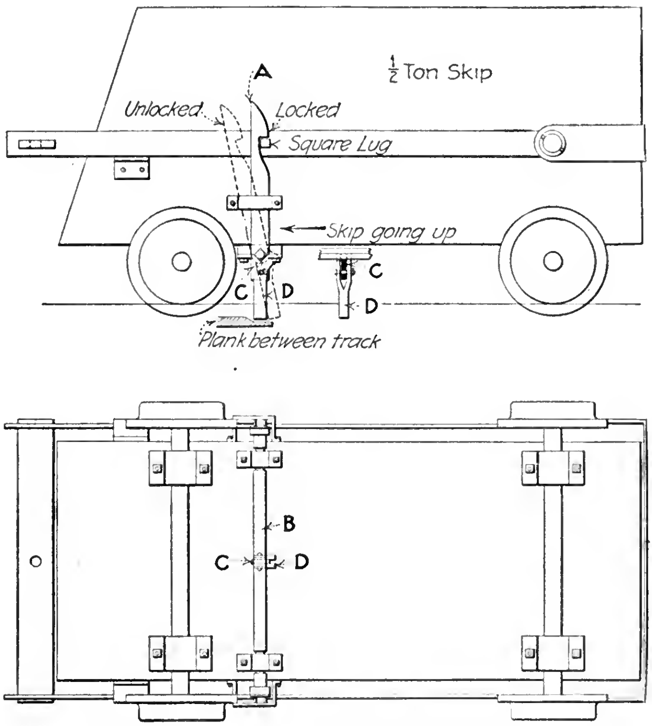


FIG. 304.—BAIL LOCK APPLIED TO SKIP.

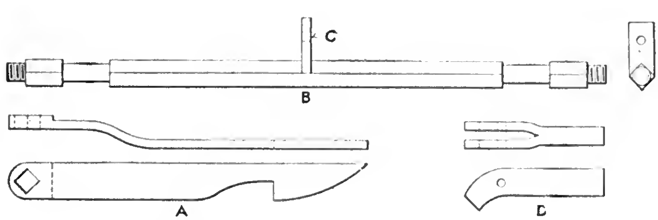


FIG. 305.—DETAILS OF SKIP-BAIL LOCK.

given, inasmuch as these would vary with every installation. The revolving shaft is made of 1¼-in. square shafting; the latches and the trigger of ⅜ × 2-in. bar iron.

Bailer for Winze (By S. A. Worcester).—The 800-gal. bailing tank

shown in Fig. 306 was designed for use in a winze of the Camp Bird mine. It is arranged to discharge into a tank placed at the south side of the winze in such a manner that two streams flow from the bottom, one on each side of the guide. The curved arm *A* projecting beyond the upper corner of the bailer is engaged by a stationary lug fixed at the discharge point in the winze. This lug forces the arm *A* inward, rotating the 2-in. shaft *B* and through the link *C* raises the rod *D* which lifts the discharge valve, thus emptying the bailer. The removable guide shoes are bolted to the sides of the bailer and are fitted with liners which can

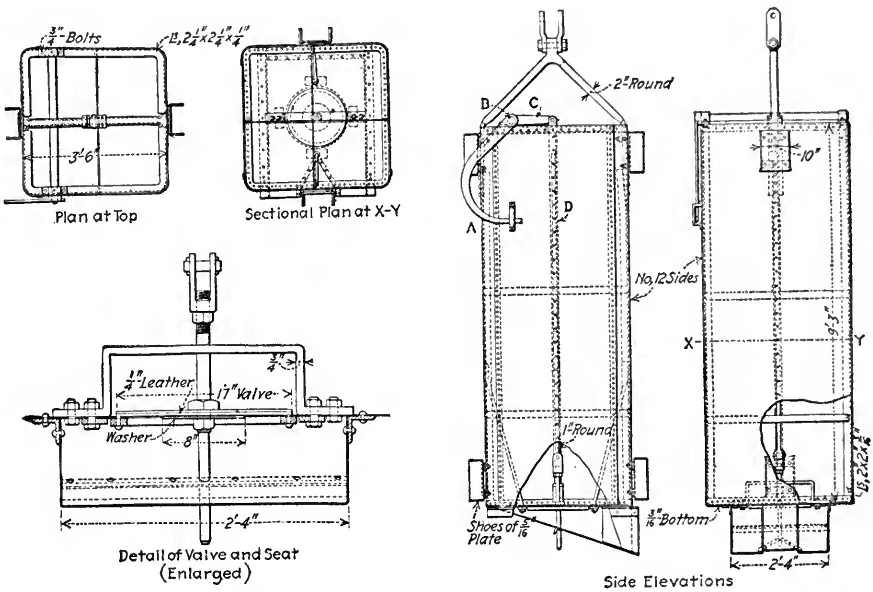


FIG. 306.—BAILING TANK USED IN CAMP BIRD MINE.

readily be changed in case of there being variations in the distance between guides. The sides of the bailer are made of light sheets well stiffened by ties and angles to prevent bulging. The bottom plate, 3/16 in. thick, is stiffened by the angles that hold the discharge spout, by the valve-seat ring, and by the two rods from the sides.

Self-discharging, Inclined, Bailing Tank (By M. G. Söhnlein).—At the Maestro shaft of the mine Socavon de la Virgen in Oruro, Bolivia, the water is bailed with tanks of 400 gal. capacity. The water contains a considerable amount of acid and sulphates, so that the tanks have to be constructed of wood. The vertical depth of the shaft is about 600 ft. and the flow of water amounts to 5500 gal. per hour. Along the sides of the shaft some clean water flows in, which is caught in a stationary

tank about 100 ft. above the 600-ft. level, whence it is hoisted to the surface. There it is discharged into another tank, from which it goes directly to the feed pumps of the boilers. The automatic discharge had to be designed in such a way that the acid water would flow into a launder at the shaft collar, and the clean water would discharge into the storage tank about 15 ft. higher. The same bailers are used for either acid and clean water and at the beginning of the shift a few buckets of fresh water are first hoisted.

The arrangement can be readily understood from Fig. 307. A is the valve hinged at H. As soon as the roller R strikes the plank P,

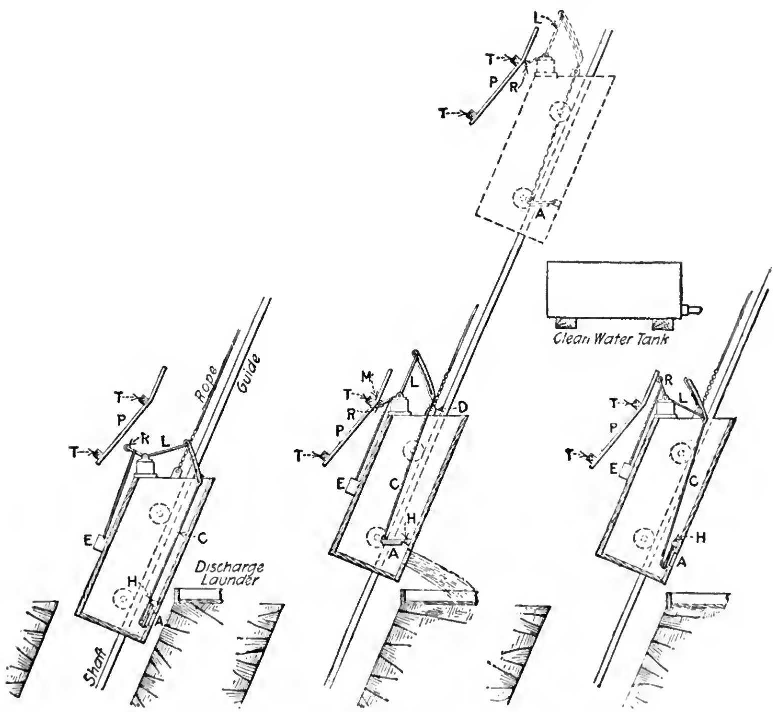


FIG. 307.—BAILER WITH TWO DISCHARGE POINTS.

which is placed a little more inclined than the shaft, the lever L is lifted and the valve is opened by the connecting rod C. At the point M the valve has reached its largest opening. The upper part of the plank, which is connected with the lower part by angle irons, is placed at the same inclination as the shaft. If the tank should be lifted a little higher, as frequently occurs, the roller travels on this part of the plank, keeping the lever in the same position as it has at M, and consequently the valve stays wide open until the roller leaves the plank, when the lever

falls back, shutting the valve. By placing another plank higher up, the discharge of clean water is led into the clean-water tank. When hoisting fresh water the engineer passes the first plank quickly so that the valve opens for only a very short time and but a small quantity of the water is lost.

When lowering again the roller has to pass the first plank. Therefore the fork at the upper end of the rod C is slotted for some distance and the

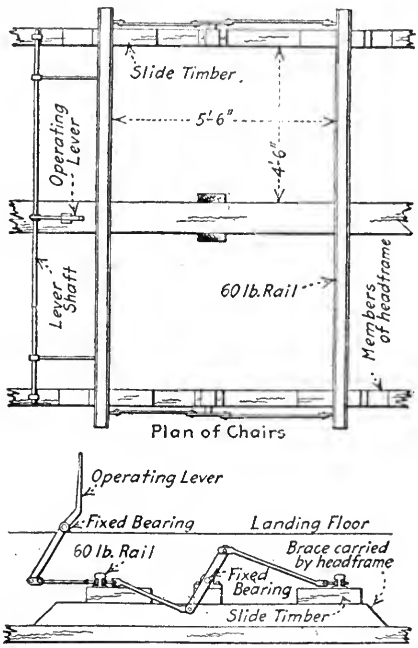


FIG. 308.—ONE SET OF SLIDING CHAIRS SERVING TWO COMPARTMENTS.

bolt which connects the rod and the lever can slide through this fork until the roller comes into such a position that it freely passes the plank, as shown. After leaving the plank the lever is brought back into its proper position by the counterpoise E. Even if the engineer should hoist the tank so high that the roller leaves the plank entirely, he can lower again and then hoist it to its proper position. Further construction details can be easily understood from the illustration. The planks are fixed to the timbers of the headframe with crosspieces T.

CHAIRS AND DOGS

Double Landing Chairs.—In Fig. 308 are shown the general features of the design of the landing chairs used at the shaft of the Desloge Consolidated Lead Co. in southeastern Missouri. The chair, which is simple,

is of the sliding and not of the leg type. It is made to serve both compartments of the shaft. The landing bars are 60-lb. rails placed head up. These slide upon blocks carried by the beams of the headframe. From the first beam the motion is transmitted by means of connecting rods and double-arm levers to the landing beam on the other side of the shaft, which in this way is thrown in at the same time. The operating lever is attached to the middle of the lever shaft so that it will be handy when landing either cage. Little handling of the two cages at the same time is necessary at the collar of a shaft, so there does not seem to be any special reason for not using a double chair at the surface. Of course,

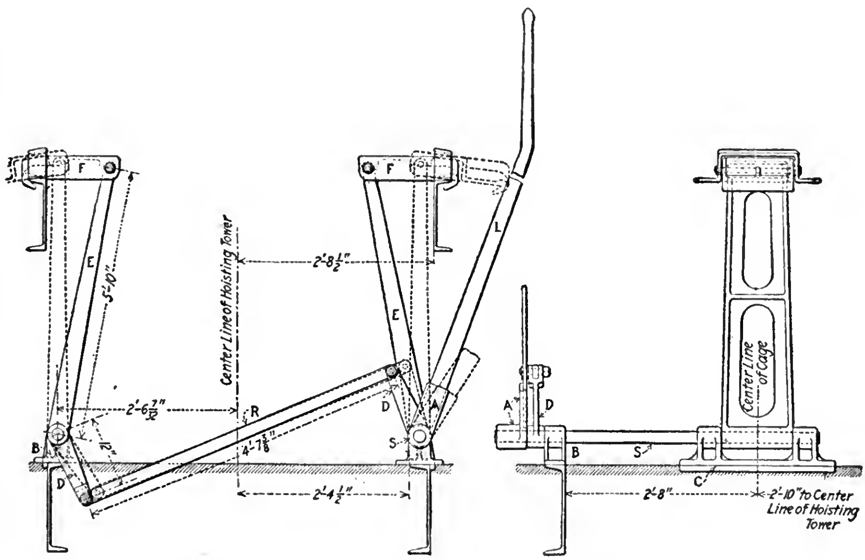


FIG. 309.—DETAILS OF CHAIRS FOR DROP-BOTTOM CAGES.

a double chair could not be used at intermediate shaft stations so safely, as it is often necessary to lower a cage past the one that is landed in the shaft.

Chairs for Drop-bottom Cages.—The details of construction of the chairs for the drop-bottom cages used at the shafts of the St. Louis Smelting & Refining Co., in southeastern Missouri, are shown in Fig. 309. The chairs are made to come out and catch the cage under the center of each side so as to lift the drop part of the cage deck, on which the car is resting, level with the outer rails of the deck before the weight of the whole cage comes on the chair. The bearings *B* and *C* are carried from a member of the headframe to which they are securely bolted. In these bearings are carried the chair shafts *S* to which are keyed three arms: The lever arm *A* to which the lever is bolted; the link arms *D* and the chair

arms E. A rod R connects the link arms of the two chair shafts together. The chair arm is a heavy steel casting about 4 ft. long. To the upper end of the chair arms are bolted the chair shoes F, which are moved out over channels riveted to the headframe at the proper height for landing the cages. On the inside end of the chair shoe is a hook that engages with the channel on which it slides and limits the inward throw of the chairs.

Skip Dog for Inclined Shaft (By Walter R. Hodge).—The dog illustrated in Fig. 310 is used in the London and Burra Burra shafts of the Tennessee Copper Co. Both these shafts dip at about 75°. The dog is

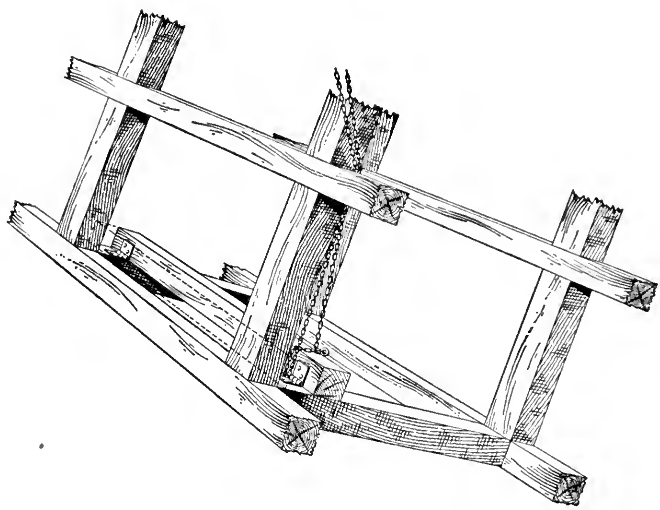


FIG. 310.—CHAIN-OPERATED, HINGED SKIP REST.

a piece of oak, 8 × 8 in., stretching across the compartment and resting on and hinged to the dividers between the compartments below the shaft station. The hinges are strongly made of ½ × 2-in. strap iron. When not in use the dog lies back and out of the path of the skip, clearing it by about 3 in. Two chains encircle the dog and are fastened to an eye-bolt through the timber. The free end of each chain reaches to the level above. When it is desired to use the dog, one of these chains is given a slight pull and the dog rolls over on its hinges and takes a position across the shaft in the path of the skip. To clear the shaft the other chain is pulled.

SKIP DUMPS

Angove Skip Dump.—The skip-dumping device for inclined tracks invented by John Angove and used at the Copper Range shafts, differs from the ordinary type of dump in having no break in the main rails.

Instead of the front wheels of the skip falling away on the bent-over main rails and the rear wheels continuing up on auxiliary rails in the original direction, the front wheels continue in the original direction and the rear wheels are elevated on auxiliary rails until the skip assumes a dumping position. A great advantage of the dump lies in the fact that it can be applied to dumping in underground bins or can be used for more than one dump in the surface bins when it is desired to hoist both waste and ore. Also, it can be arranged to intercept a skip falling back from a broken rope after an overwind.

Referring to Fig. 311, the skip is shown in dumping position and the action of the auxiliary tracks is apparent. They are brought down outside the main rails and the wider treads of the rear wheels are engaged by them while the narrower front wheels pass through. The connecting point *B* of the auxiliary outside rails can be permanently fastened to the lower track so that any skip passing is forced to dump, or it can be hinged at *A*, as shown in the drawing, in which case the dump can be made operative or not, as desired. When open, as shown in the dotted position, the skip is allowed to pass through. The movable point *B* has a stud *C*, which plays in a slot in the rocker-arm *D*. This rocker-arm and another one *E*, are attached to the rocker-shaft *F*. By depressing *E* through the connections shown, the switch point *B* is opened, and by raising it is closed. The two counterweights *G* and *H* make the movement of the operation relatively easy. Inasmuch as no shafts can extend across the track where the skip runs, a certain portion of this mechanism must be repeated on the other side.

At the upper part of the auxiliary tracks, a pocket is cut in the supporting plate, which is designed to catch the rear wheels of a skip and prevent its descent into the shaft if the rope should break for any reason. The guard rails *K* are provided to keep the rear wheels in their proper position. At the top of the chute, in such a position as just to catch the mouth of the dumping skip, are the rollers *L*, which prevent the front end from falling down and binding against the chute. When a long skip is used, the hanging opposite an underground dump may have to be cut away to get room.

Adjustable Skip-dump Plate (By W. C. Hart).—At the Wakefield iron mine in Michigan it was found desirable to start operations with 3-ton skips in the vertical shaft, these being most suitable for the initial hoisting equipment and being large enough to handle the development production. There was, however, a possibility of 5-ton skips being used at a future stage of operations and it was decided for this reason to design dump plates which would allow changing from the 3-ton to the 5-ton skips with a minimum of delay and labor. Fig. 312 shows a pair of combination plates. The plates, as shown, are made up for 5-ton skips. To get them

ready for 3-ton skips it is only necessary to unbolt the roller hub *A*, and bolt it into position shown by dotted lines *B*; unbolt the 13½-in. length of angle between *C* and *D*; unbolt the manganese point *C* and bolt it in position shown by dotted lines at *D*. The change can easily be made in an

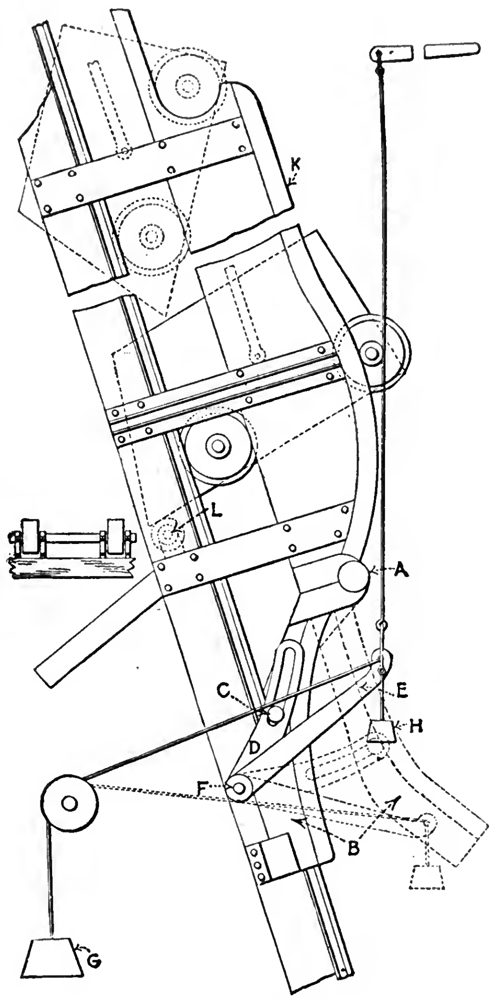


FIG. 311.—SKIP DUMP FOR INTERMITTENT SERVICE.

hour, between shifts. The design was worked out graphically on a large scale, so that there is perfect coördination between the 3-ton skips now in use and the plates. The guide angles at the tops of the plates provide for overwinding at the angle of discharge of the skip, and there is no possibility of the ore dropping back into the shaft. Runners are built in

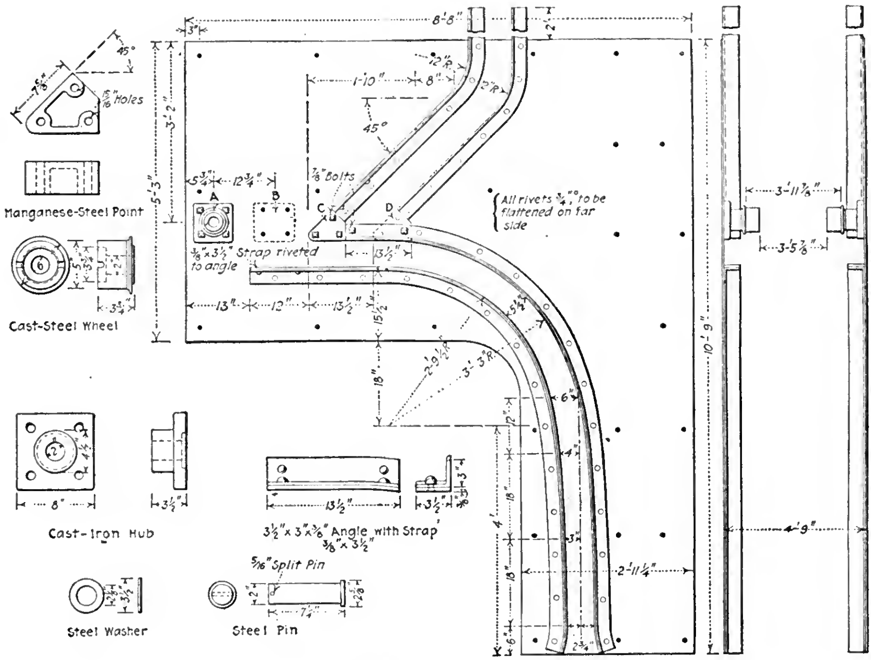


FIG. 312.—DUMP TO ACCOMMODATE EITHER 3-TON OR 5-TON SKIP.

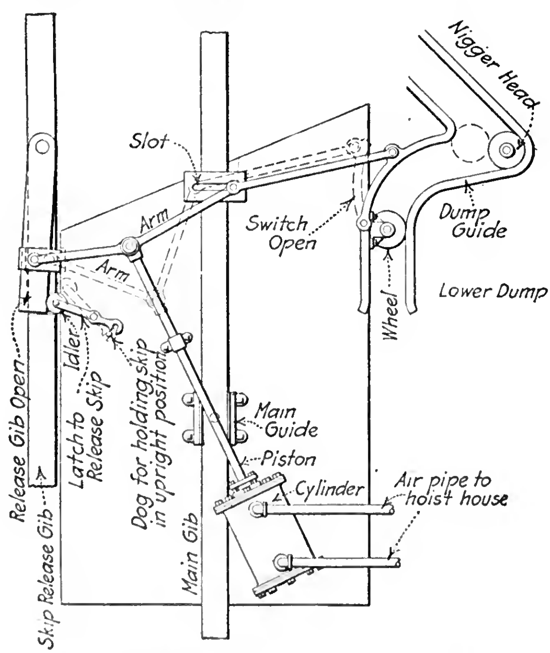


FIG. 313.—ARRANGEMENT FOR CUTTING SKIP DUMP IN OR OUT.

the headframe at the upper end of the plates, as a continuation of the guide angles of the plates, so that the skip will remain at the angle of discharge for an overwind of 10 ft.

Compressed-air Dump Control (By J. R. McFarland).—The Giroux Consolidated Mines Co. at Kimberly, Nev., had to make arrangements for hoisting both ore and waste with the same skip and without loss of time in changing from one to the other. It was desired to avoid wasting time from having the skip dump into one chute and having a man climb up into the headframe and adjust a gate so as to direct the material into the ore bin or waste bin as the case required. Therefore two bins were built, the lower for waste and the upper for ore. In operation, if a car of waste was hoisted, the engineer, by the operation of a lever at his side, controlling air to the dump machinery, closed a switch in the lower dump guide and released a latch on the skip, allowing the skip to swing at its center on the main guide. Thus the waste was dumped into the lower bin. If it happened to be a car of ore, by the operation of the same lever, he opened the switch in the lower dump guide, allowing the guide wheel to pass through, and at the same time the release gib was held closed so as to maintain the upright position of the skip. The skip thus passed through the lower dump and dumped the ore into the upper bin. The arrangement of the dump mechanism is shown in Fig. 313.

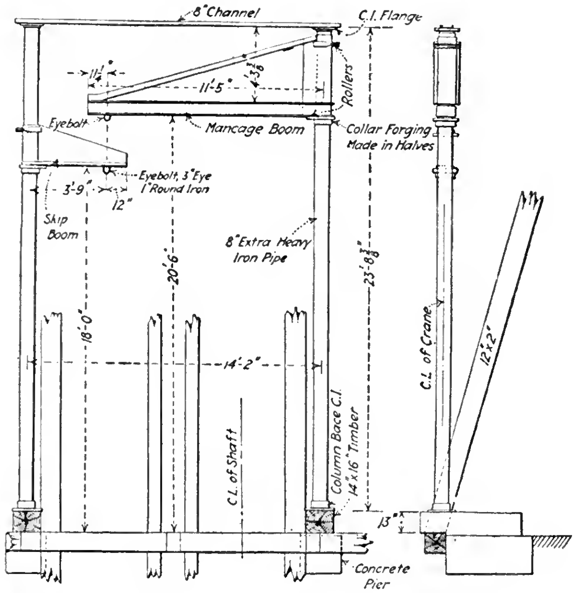
TRANSFERS

Skip and Man-cage Transfer (By Walter R. Hodge).—The device herein described is in use at the London and Burra Burra shafts of the Tennessee Copper Co., Ducktown, Tenn. At these shafts men are handled in one only of the three compartments. The device is used for swinging skips and man cages in and out of the shafts when it becomes necessary to change them.

The transfer, Fig. 314, consists of two vertical columns of extra-heavy 8-in. iron pipe, on each of which swings a boom. These booms are of unequal length and are so arranged that an eye-bolt near the end of each may be swung over the center of the compartment in which the men are handled. In each eye-bolt are hung two chains with hooks at the end. These hooks engage holes in the sides of the man-cage and of the skip, near the top of each. The booms are built of ¼-in. plate and structural steel. They rotate about the columns on vertical rollers placed at the bottom of the boom on the side of the column from which the boom extends, and on the other side at the top of the boom. A collar bolted below each boom prevents it from slipping on the column. About the column at the bottom is shrunk a heavy cast-iron base which is in turn bolted to a concrete base or pier. Across the top the two columns

are joined by an 8-in. channel brace, which is frequently used as an aid in loading material in the skip to be taken underground. The whole structure is stayed by four steel guy-ropes fastened to convenient "deadmen."

In operation, when it becomes necessary to exchange skip for man cage, the skip is hoisted into position near the long boom, the bail blocked



FIG, 314.—COLUMNS AND BOOMS FOR CAGE AND SKIP TRANSFER.

with a length of drill steel to prevent its dropping down and the chains attached by means of their hooks to the holes in the sides of the skip. The cable is then slacked off so that the skip swings on the short boom. The bolt is removed from the cable clevis and the skip swung out of the shaft by means of a rope reaching the ground and handled by the dumpers. The man cage is swung into position on the rails by means of the long boom on which it is already suspended, the cable is attached to the bail, the cage hoisted and the chains cast off. The whole operation occupies about two minutes.

Double Skip-changing Carriage (By William Hambley and Albert E. Hall).—One of the most frequent time losses in hoisting is that due to changing skips for one reason or another, often for repairs to the skip in use. The operation of changing a heavy skip without any device other than a cradle to span the top of the shaft is slow and uneconomical A time-saving device for this operation is shown in Figs. 315 and 316. It is a carriage designed to hold two skips, the dimensions given being subject to alteration as the case demands.

In the installation considered, there are two shafts connected by a track, which also runs to the blacksmith shop, the machine shop and the repair yard, Fig. 315. The skip carriage is designed to run on this track, as is also a car to carry drill steel from the blacksmith shop to the shafts.

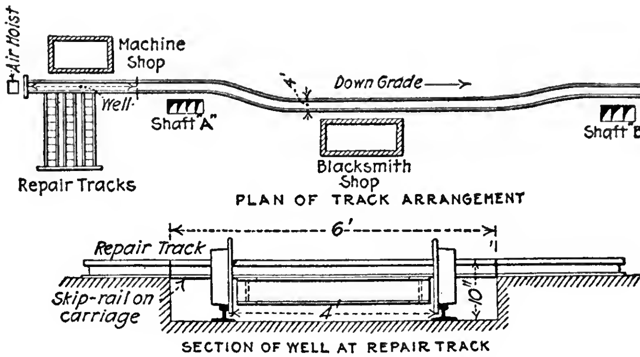


FIG. 315.—PLAN OF SURFACE ARRANGEMENTS AND REPAIR WELL.

A small, stationary engine, operated by air, pulls the vehicles by means of a light cable. There are two points requiring care in laying the track: First, it must be placed far enough back from the shaft to allow the nose

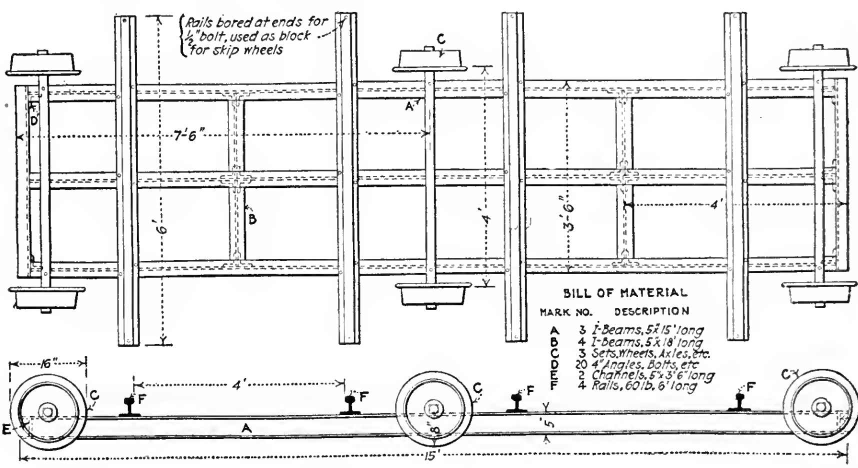


FIG. 316.—DESIGN OF THE DOUBLE CARRIAGE.

of the skip to clear the timbers supporting the shaft guard rails; the nose of the skip will, of course, project some distance over the skip carriage if the shaft is inclined. Second, the track at the shaft should be laid flush with the ground to avoid the danger of anyone's tripping on it and plunging down the shaft. The carriage itself, Fig. 316, is made as low as pos-

sible, so that the center of gravity will be low, thus reducing the tendency for the carriage to tip up as a skip is loaded on it or taken off.

A skip in good condition is always kept on the carriage. When it is necessary to change skips, the carriage is run to the shaft, the skip is hoisted a short distance above the surface and the spanning rails thrown across the shaft. The skip is lowered and run on the empty half of the carriage. The carriage is then advanced a few feet, the new skip run off and connected to the rope. This operation takes but a short time and is much easier than the old method, requiring eight or ten men with bars. When the skips have been changed, the carriage is run to the repair yard, the old skip unloaded and a new one put on ready for another emergency.

Skip Transfer at Incline-shaft Collar (By Arthur C. Vivian).—At the conglomerate shafts of the Calumet & Hecla, in Michigan, the skips and man cars when not in use are stored on horizontal tracks in a detached building, about 30 ft. from the shafthouse proper. The conveyances are transferred from the incline tracks of the skip road to the horizontal surface tracks over short pieces of track curved in vertical section, which may be swung into place for that purpose. When not in use these sections of auxiliary track must be out of the way of skips in the shaft, and two methods of disposing of them are employed, the choice of which depends upon the weight of the skip used.

The simpler and more convenient of the two methods may be used only with a small skip, this for the reason that the two rails have no lateral bracing except that afforded by the makeshift wooden prop, and a heavy skip would be likely to spread the rails in passing over them. This method is shown in Fig. 317, at1, in plan only, but a side view would correspond except in details to the side view of the other method shown in 2. Rails of about 90 lb. weight are used, and they are not reinforced or connected with each other in any way except near their junctions with the horizontal track, where they are both pivoted to a switch bar so that their upper ends may be swung aside horizontally to clear the skip. The switch is used to make connection with either of two tracks, one holding the skip, the other the man car. The method is obviously applicable only to a single-skip-road shaft, as there would not be room between the adjacent, inside rails of two-skip roads to allow the auxiliary rails to be swung aside. A casting riveted to each rail near its upper end projects below it and engages a corresponding slot in the track stringer to hold the rail in place; a sprag is braced against each rail from the sides of the shafthouse to prevent the rails from spreading. The rails can be thrown in and out of position easily and quickly by one man, and it is unfortunate that this simple method is not of wider application.

Where heavier skips are used the second method is employed, and this conforms in principle to that used at most of the shafthouses of the

district. The two rails are connected rigidly by cross-braces, and they are hinged at their junction with the horizontal track so that their upper ends may be lifted high enough to clear the skip. The method is illustrated in plan and side elevation in 2, Fig. 317. Most shafthouses are equipped with a pony winch for handling timber and this may be utilized

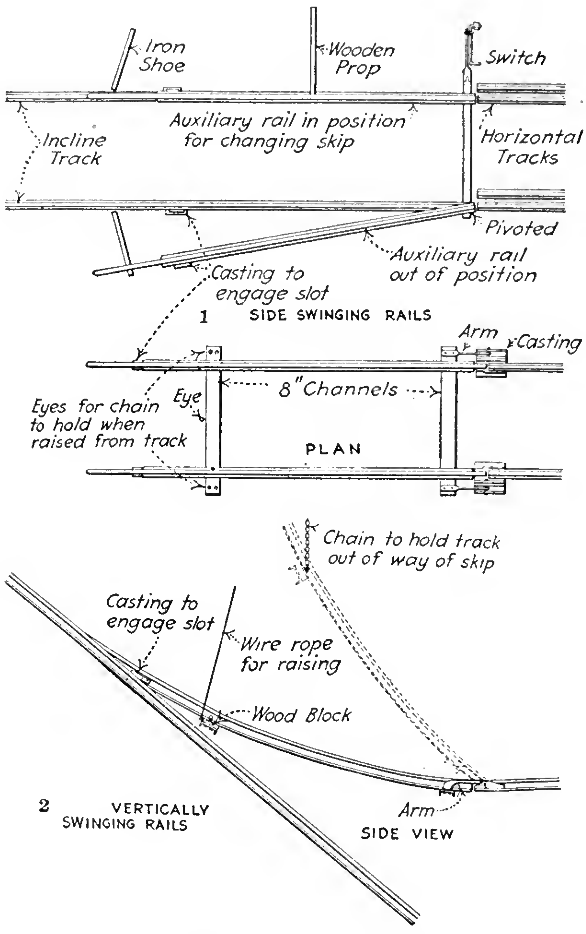


FIG. 317.—ARRANGEMENTS FOR TRANSFERRING VEHICLES.

in lifting the auxiliary rails, or in the absence of this accessory they may be lifted by hand. When not in use, they are suspended by chains hung from the roof. The illustration shows a heavy rail without any reinforcement in the way of backing, but at other mines it is usual to mount the rail for part of its length on a wooden stringer. On steeply inclined tracks, such as those at the Allouez shafts and Nos. 3 and 4 shafts of the

Ahmeek, where the track is inclined at 80°, this reinforcement takes the form of guides for the skip wheels, so that they will not leave the track on the sharp vertical curve used.

Device for Holding Skip Rope (By L. Hall Goodwin).—The device used in the shafthouses of the Quincy Mining Co., Hancock, Mich., for holding the skip rope while one conveyance is being substituted for another on it, has advantages over the cruder methods generally employed.

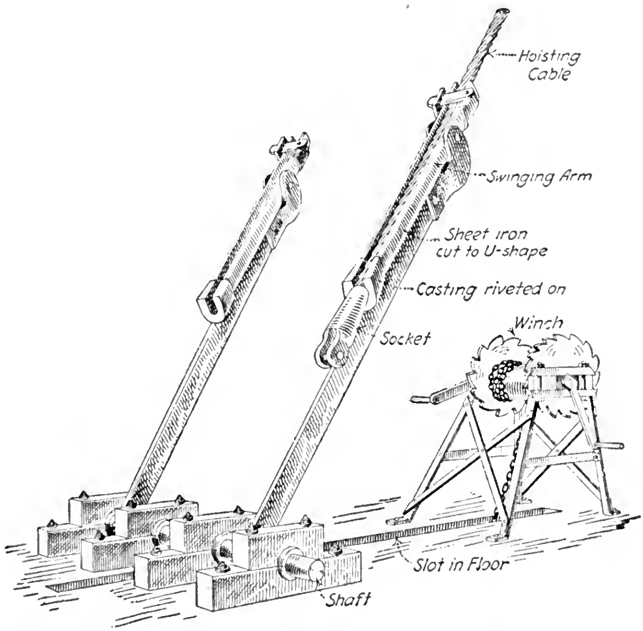


FIG. 318.—ROPE-HOLDING APPARATUS.

The arrangement consists of three arms rigidly fixed to a 4-in. round iron bar. The two arms shown in Fig. 318 carry the holders for the ropes, and each swings in an arc in the vertical plane which contains its respective rope. The third arm lies midway between the other two and extends beneath the concrete floor, a slot being left in the floor to allow the arm to swing; it carries a counterbalancing weight and also serves as a lever for rotating the bar. The device is set at such a height in the shafthouse relative to the cranes on which the skips and man cars are stored that when one of them is swung into position over the track it will be at the right height for adjusting the clevis to the rope socket. When a conveyance is to be changed, the holder is swung into position by rotating the arms until the holder engages the hoisting rope. The rotation is accomplished by means of a small hand winch which winds a chain attached to the counterbalance arm. On either end of the winch

drum is a ratchet wheel, one of which holds the arms when in position, the other when out of position. The illustration shows the manner in which the holder catches the rope. For convenience one holder is shown empty, the other engaging a rope. As a matter of fact the arms are in the position assumed when they are out of service; they are swung up when they are to be used.

BUCKETS

Joplin Ore Buckets.—In the Joplin district of Missouri practically all the ore is hoisted in buckets. These buckets, or cans or tubs, as they are commonly called, are made with straight sides. The ears are welded to straps instead of being riveted to the sides. The buckets are used without a crosshead to guide them through the shaft, and so have to stand much rough usage. They are made with a bottom flange, Fig. 319, about 1½ in. deep so that they will rest flat upon the tub cars. On the bottom there is a cross-strap riveted to another strap inside. This distributes the strain of the pull from the bottom ring while dumping the bucket at the surface. The side straps also are riveted to stiffening straps on the inside while the ears are of 1¼-in. Norway iron welded to the 2½ × ½-in. side straps. The bail is of 1½-in. Norway iron and is V-shaped with a slight curve at the apex for the reception of the hook. This bail is made with the hooks turned in so as to be less liable to catch on timbers during hoisting.

Usually the side straps come down and turn up so as to hook over the flange at the bottom, thus throwing the weight of part of the load on the hook as well as on the three rivets by which the side straps are attached to the bucket. But in the case of the buckets at the Oronogo Circle mine the bottom straps run up the sides about a foot and are overlapped by the side straps to which they are riveted, as illustrated. In this way the strain of dumping is taken from the bottom, which already takes the most of the wear and tear of usage, and is thrown on the sides. A slight drawback is that the whole weight of the load is thrown on the three rivets of the side straps. At the Underwriters mine, commonly called the Yellow Dog, an extra piece of iron is put in under the bottom strap, as shown, to keep the ring from punching a hole through the bottom of the bucket.

In most of the buckets used in the Joplin district the bottom is only a sheet of No. 8 iron. To protect this from wear the shovelers are made to throw in fine dirt at first, and then after a protecting layer has been formed on the bottom, the boulders are loaded. In the case of the Federated Zinc Co. and a few others, tubs with wooden bottoms are used. This wooden bottom of 2-in. oak is protected from the cutting of the boulders by a plate of No. 10 iron, and seems to be serviceable. The flange at the

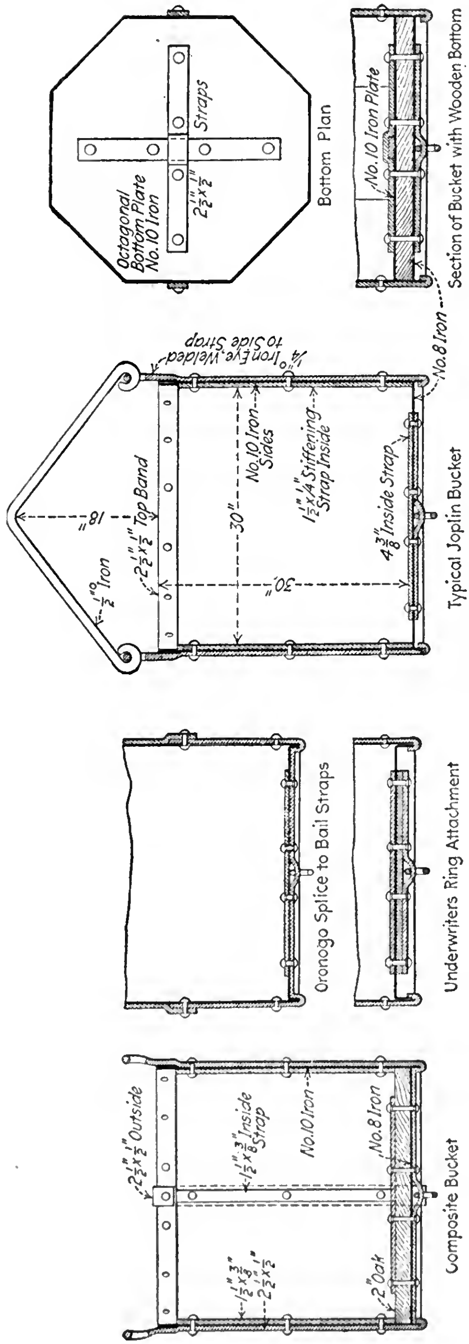


FIG. 319.—TYPES OF ORE BUCKETS USED IN JOPLIN DISTRICT.

bottom of the bucket becomes battered in the course of time. In one instance instead of flanging the bottom plate, buckets were made with an angle-iron ring riveted to the sides at the bottom. The bottom plate then rested on this angle, but such buckets are much more expensive to make than are those of the flange type.

Owing to the fact that the hoisting is done without a crosshead, the buckets often catch on the cribbing timbers. On that account the top of the bucket is reinforced by a band of $2\frac{1}{2} \times \frac{1}{2}$-in. iron. A few buckets are built with side stiffeners put in half way between the side straps. These run vertically from top to bottom of the bucket and are turned over to form a hook over the top band as well as at the bottom flange. As these extra side straps are riveted to the top band there is less danger that the bucket may bend out of shape when it catches on the timbers in the shaft. But the main advantage is that the bucket, in landing on the plat at the bottom of the shaft strikes on these side straps; consequently the flange is less liable to be bent.

In many instances, the bottom strap is extended clear across the bottom and caught in under the turn-up of the ear straps. This seems to be the best way of putting on the bottom strap, since part of the strain is thrown on the side straps while the bucket is being dumped and taken off the bottom strap, but it is not commonly done as the bucket is slightly harder to make. Probably the best bucket is obtained by toeing the bottom strap under the hook of the ear straps, using a wooden bottom, and putting in the extra side straps already mentioned. But there is the drawback that this makes the bucket somewhat heavier and more awkward for the hooker to handle when he changes buckets at the bottom of the shaft. Moreover, it is somewhat more costly in upkeep. In case a wooden bottom is not used, it is well to put in the extra strap so as to protect the bottom from wear by the ring. Such a bucket combining the good points of several of the "tubs" in use in the Joplin district is shown in the illustration.

These tubs or buckets are made in certain standard sizes. The largest buckets used in the Joplin district are those at the Davey mines of the American Zinc, Lead & Smelting Co. These are 34 in. in diameter and 34 in. high. They hold as loaded about 1200 lb., but at the rating placed on buckets in the district these large buckets would be called 1600-lb. cans. The most commonly used sizes are the 30 × 30-in. buckets, with 30 × 28-in. and 28 × 30-in. close seconds, the diameter being given first in all cases. These are called 1200-lb., large 1000-lb. and small 1000-lb. cans, respectively. Other sizes are 26 × 28-in. called 850-lb., 24 × 28-in. rated at 750-lb., and 22 × 26-in. rated at 500-lb. In some instances 28 × 32-in. and 30 × 32-in. as well as 32 × 34-in. tubs have been used.

None of these buckets holds what it is rated at, for they are rarely

filled more than four-fifths full. In mining, the ore breaks in many boulders, especially in the case of the stope holes, so that there is much empty space in the buckets. Indeed, it is necessary to keep a close watch upon the shovelers to prevent them from building "windies," as they are called in the district, by piling the boulders into the tubs in such a manner as to leave a large percentage of the tub unfilled. The larger the diameter of the bucket, the more difficult it is to build these "windies;" hence it is better to use either a 30 × 30-in. or 30 × 28-in. than a 28 × 30-in. bucket, especially when mining the deposit by a "stope." The real amount that these different buckets hold depends upon the richness of the ore which to a large extent governs the fineness to which the ore breaks, and also upon whether the ore comes from a stope or a heading. The 30 × 30-in. cans which are rated at 1000-lb. capacity throughout the district have been found, in many instances when their contents were weighed, to hold about 850 lb. on an average when loaded with dirt broken fine from a heading, while the buckets that were loaded from the stope piles weighed only 800 lb. Many of the operators in the district make such allowances in calculating their yield and their costs, but some use the rated capacities.

Bucket for Lowering Drill Steel (By Evans W. Buskett).—In the practice of underhand stoping in the sheet-ground mines in the Joplin district, drill bits sometimes reaching a length of 16 ft. are used. To send such drill steel the special steel can, shown in Fig. 320, was devised. The bucket proper is about 24 in. in diameter and 30 in. deep. In the center is a cylinder about 9 in. in diameter passing through the bottom of the can and braced at the top as shown. This cylinder can be made to suit the length of steel used. The sketch shows a cylinder for 16-ft. drills. In use the can is set in a hole in the platform at the collar of the shaft and the steel is placed in it. The longer pieces are placed in the cylinder while the short ones are placed in the compartments made by the braces. When the can reaches the bottom of the shaft it is laid on its side and the rope detached.

Man Platform and Bucket Crosshead (By E. M. Hobart).—In many isolated mines where hoisting buckets are used, safety crossheads cannot be obtained on short notice. In such cases the crosshead shown in Fig. 321 has advantages for temporary use. One occasion arose when it was necessary to demolish a wooden headframe over a 1400-ft. shaft, in order to erect a steel structure in its place and at the same time retimber the shaft. One sheave was placed on the center-post section of the old frame, which sufficed as a temporary frame with the addition of a few braces, but was not strong enough to support a cage. The addition of a platform, as shown in the illustration, to the crosshead, solved the problem of lowering men. The platform accommodated six men, and the

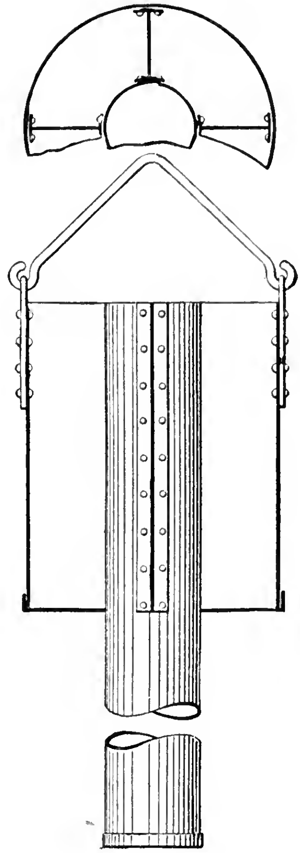


FIG. 320.—DRILL-STEEL BUCKET WITH LONG CENTRAL SECTION.

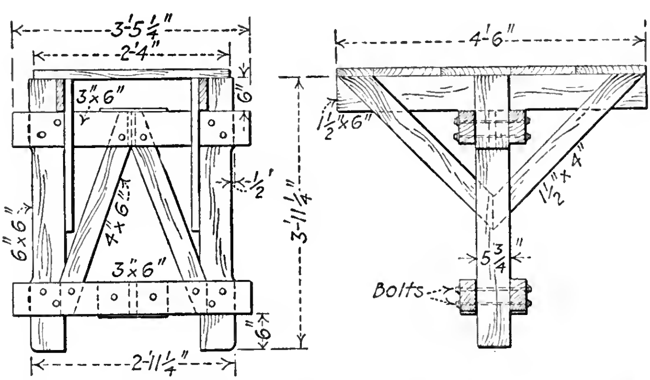


FIG. 321.—BUCKET CROSSHEAD WITH PLATFORM FOR LOWERING MEN.

bucket was attached to a chain below the crosshead sufficiently long to lower 15-ft. timbers.

Rapid and Safe Bucket Connection (By Joseph Goldsworthy).—Fig. 322 shows a connecting pin for sinking buckets, which, although not new, has proved safe, quick and simple in practice, and which obtains the confidence of the miners. No dimensions are given, as these will vary in almost every case; for a $\frac{3}{4}$-in. rope, there would be used a $1\frac{1}{8}$-in. pin

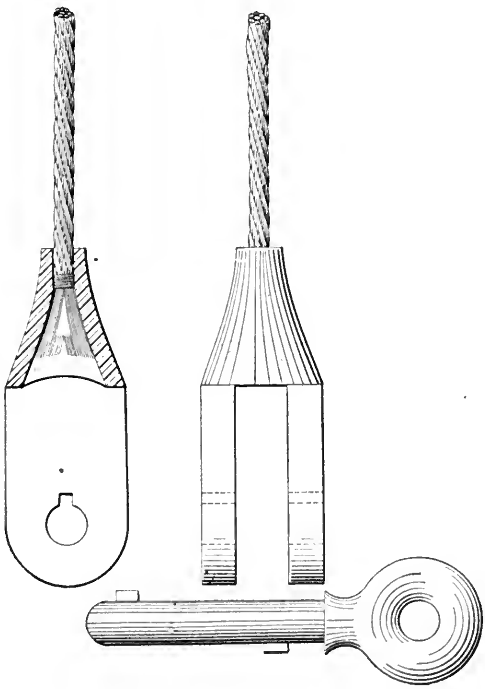


FIG. 322.—LUG PIN FOR CLEVIS OF BUCKET.

with two lugs, which are $\frac{3}{8} \times \frac{3}{8}$ in. by $\frac{1}{2}$ in. long. A slot is cut in the top of the pin hole through the jaws of the clevis socket, as shown. The slots are just large enough to take the lugs with sufficient clearance so that the pin can be quickly placed in position. The distance from inside to inside of the lugs should be slightly greater than the distance from outside to outside of the jaws of the socket. It will be seen that the front lug is on top of the pin, and the back lug on the bottom. The pin is inserted by pushing the top lug through both slots, giving the pin a half turn so as to bring the last lug on top, pushing this lug through one slot and giving the pin another half turn, bringing it to the proper position, as shown. It will be readily seen that the chances of the pin's working out are remote;

it would have to make a half turn to get the back lug into line with its slot, then travel endwise to get this lug to the outside of the jaw; at this point the front lug would be on the bottom of the pin, and would be holding, so that the operation would have to be repeated before the pin could fall entirely out. The pin is simple, without working parts to get out of order, or to clog with dirt, and in changing buckets, connections are made as quickly with this device as with any other.

BUCKET DUMPS

Joplin Method of Dumping Buckets.—The following method of dumping buckets is practised in the Joplin district when tramming the ore on the surface to a bin near-by, as when the ore is being hand jigged, or for dumping buckets of waste as in shaft sinking. The shaft is straddled by truck tracks which are laid with a wide enough gage so that the bucket will come up between the rails without hitting. On the truck is nailed a 2-in. cleat against which the front side of the bucket rests when it is landed on the car, while at the back end is a 4 × 6-in. timber also nailed to the deck. On top of this back piece the rear end of the bucket rests in a tilted-forward position. The track, when tramming to a near-by mill, generally has a considerable grade so that the trammer can ride the truck out to the dump. He gets up such speed to the load that when it strikes the cribbing of the dump the momentum causes the bucket to tilt forward. During this tilting the bucket is generally guided by the man, while at the same time he gives the bucket a slight push forward just as it strikes the dump, so as to aid in unsetting in case the momentum of the load is not enough. The buckets on which these dumps are worked generally are not larger than about 22 × 30 in. A tall can, of course, works better than one with breadth equal to height.

The dump is a crib built up on the bin or the tracks to approximately the height of the top of the bucket car, about 15 in. This crib is securely tied in place so that the shock of the constant bumping will not move it and the crib timbers are securely spiked together so as to stand the work. The crib is about the same width as the track, or several inches wider than the bucket, so that provision is made for side swing during the tipping of the bucket. In the direction in which the bucket dumps, the crib is somewhat greater than the width of the bucket, yet less than the diagonal of the longitudinal section of the bucket. Consequently, in dumping, the top of the rim of the bucket strikes on the far side of the crib while the lower part of the side of the bucket is resting upon the near side of the crib. This throws the bucket into an inclined position and the rock or ore slides out aided by its momentum. Owing to the roughness of the crib walls it is impossible for the bucket to go down through, although

the section of the dump crib is greater than the bottom section of the bucket.

In some instances these dumps are rigged so as to allow side dumping of the car. The cribbing is then built alongside the track, with a bumping block fixed across the track to stop the car even with the dump crib. But in this form no aid comes from the momentum of the load and the block on the back of the car truck has to be made so as to throw the bucket into practically a balanced position with the top pointing to the side on which the can is to be dumped; then with a slight push the trammer can dump the bucket. This dumping system is surprisingly simple and effective. Flying dumps are generally made, since they are easier on the man, and seldom, if ever, does the bucket fail to take the dump properly. In getting the bucket back on the car, all that is necessary is to tip it straight back the way it came over and slide it back a few inches so that its front is behind the forward cleat.

Device for Bucket Dumping (By J. R. McFarland).—An ingenious dumping arrangement is employed at a Nevada mine. Hoisting is conducted with a bucket, and it is customary to keep the collar of the shaft closed with hinged doors, except when the bucket is passing through. To manipulate these doors and dump the bucket, either necessitated the employ of an extra man at the shaft, or made a large amount of additional work for the hoistman in running back and forth between the hoist and the shaft. Four round trips were required on his part for each hoisting trip, as follows: To shaft to open doors, to engine to raise bucket, to shaft to insert dumping hook, to engine to dump, to shaft to detach hook, to engine to lower bucket, to shaft to close doors, to engine to await next trip. This extra work was eliminated by the use of the devices illustrated in Fig. 323.

On receiving the signal to hoist, the engineer releases the lower rope. This allows the counterweights to pull the chute up and away from the shaft, the chute being pivoted at *A*. The chute pulls with it the sliding rods *B*, just above the collar, which by means of connecting arms opens the doors *C*. The bucket is then hoisted until the crosshead passes above the pins set in the guides at *D*. The upper rope is next pulled by the engineer, throwing in the pins against the force of the springs shown in the drawing. The bucket is lowered until the crosshead rests on the pins and holds them in position. The lower rope is pulled, bringing the chute into position under the bucket and closing the doors. The chute, in swinging over, catches the tail chain of the bucket in a notch in its upper end. The bucket is lowered and is dumped as shown, the tail chain preventing it from sliding in the chute without dumping. The bucket is hoisted, and as soon as the crosshead is lifted from the pins, the springs force them back. The lower rope is released again, swinging the chute out of the way and

opening the doors. The bucket and crosshead are lowered and the doors closed. Different arrangements of the counterweights from that shown can, of course, be used.

Automatic Bucket Tipple (By D. A. Cavagnaro).—The Channel Mining Co., of San Andreas, Calif., has installed an ingenious device

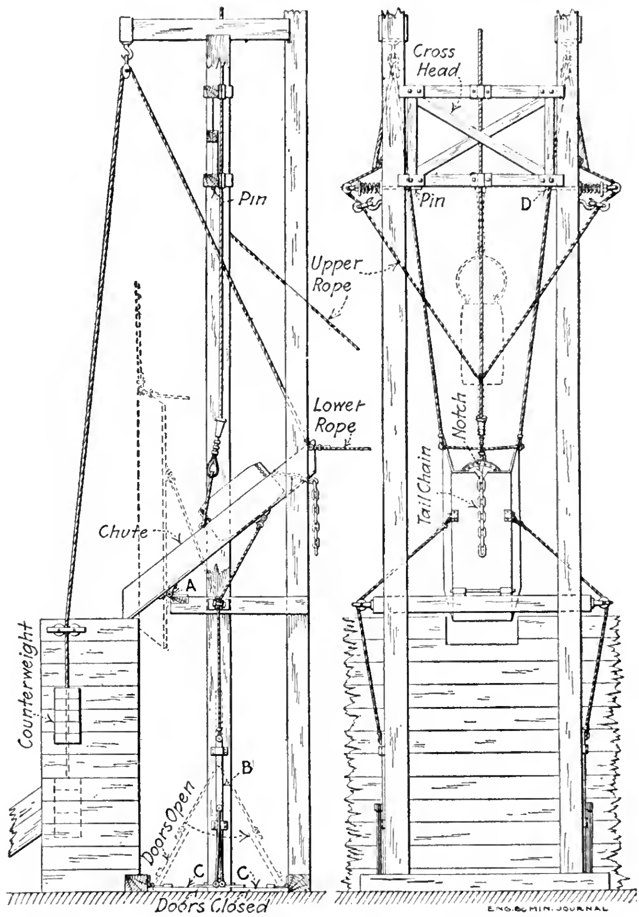


FIG. 323.—ARRANGEMENT FOR DUMPING BUCKET WITHOUT LEAVING HOIST.

which permits the hoisting engineer to dump a hoisted bucket without leaving his engine. Referring to Fig. 324, the lower side view shows the tipple with the bucket in place. The upper side view shows the dumping position. The top plan shows the bucket in the tipple but not the catches or the rotating control. The back view shows the tipple without the bucket.

The tipple itself is built around four upright pieces, *P*, which are bev-

eled to permit the entrance of the bucket. On the sides $\frac{1}{4} \times 6$-in. iron straps I are bolted to the uprights. The top of the back is closed by two 2×12-in. planks L. The iron hook B is bolted to these and also any

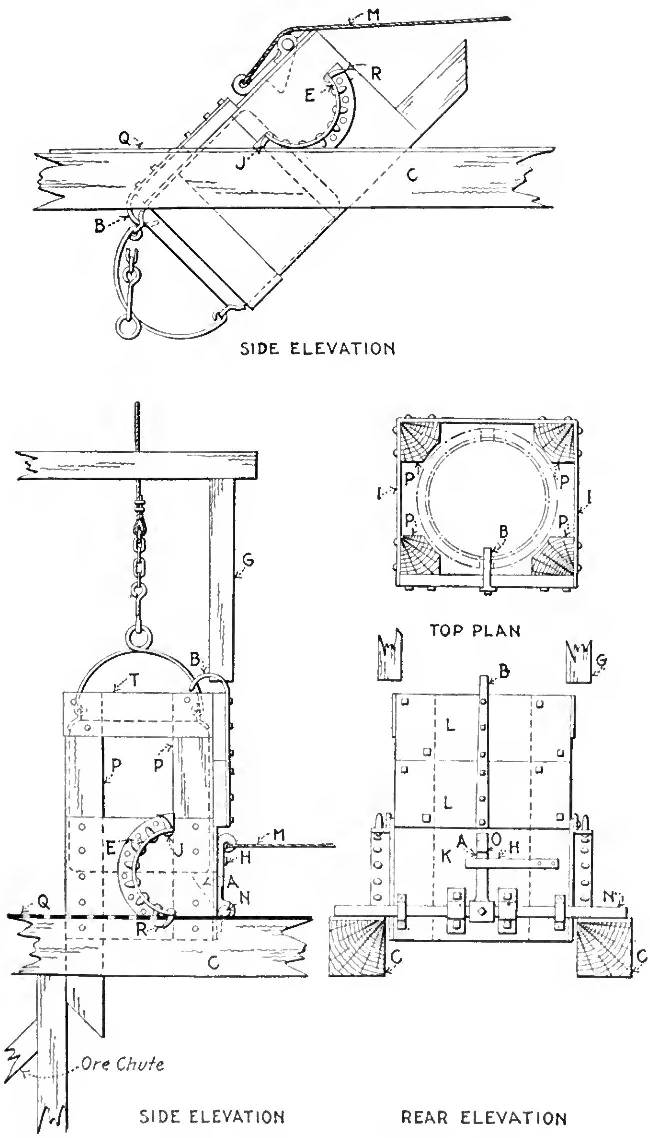


FIG. 324.—DEVICE FOR DUMPING BUCKET.

weights which may be necessary to hold the tipple upright. The bottom is closed by a $\frac{3}{8} \times 24 \times 30$-in. iron sheet K, through which the latch A

works. The latch is held inside the tipple by the spring *H* and from its upper end *O* a rope *M* leads to the hoist. The latch *A* works on the 1¼-in. round iron *N*, which extends to the sides far enough to rest on the timbers *C*, and thus supports the tipple. To the sides of the tipple are bolted iron plates which carry two half-circles *E* with a radius of 18 in., made of 2½ × 2½-in. angle iron, and in these are set a number of teeth. These teeth are spaced 3 in. apart and are 1 in. from the outer edge. The inner teeth are about 1 in. long, but the outer two teeth, *J* and *R*, are curved, about 3 in. long, and riveted in square holes to prevent turning. These teeth mesh into holes in the horizontal plates *Q*, fastened to the timbers *C*. The tipple at rest pivots on the teeth *R*, and the half circle is so set that the pivot point is about 6 in. behind the center, which arrangement causes the forward dumping movement. In operation the bucket is hoisted into the tipple by the engineer and forces back the latch *A*, which then springs in again and catches the bottom of the bucket. When the rope is slacked, the device tips forward by its own weight, the operation being controlled by the teeth of the half-circle *E* meshing into the holes in *Q*, and the contents of the bucket discharge into the ore chute. The bucket is prevented from sliding out while tipped, by the hook *B*. When empty, the engineer pulls the tipple back to an upright position by means of the hoisting rope and by pulling the rope *M*, releases the latch *A* and permits the bucket to descend. As the tipple is not fastened to the timbers *C*, it could be pulled through by the bucket but for the timbers *G*, which catch the back edge.

Bucket-dumping Device (By Harold A. Linke).—A safe and serviceable device for dumping buckets at the collar of a shaft during sinking operations is constructed as follows: A chute *A*, Fig. 325, is built of 2 × 10 plank, the width depending on the size of the bucket used. The sketch shows a convenient size for use with a bucket of 9-cu. ft. capacity. The floor is laid diagonally and doubled, one set of plank crossing the other at acute angles, as shown at *F*. The sides, two planks high and of single thickness, are reinforced with cleats and strap iron. The chute is hinged at *B*. At the center of the chute at the upper end a slot *H* is cut about 12 in. long and 2¼ in. wide, reinforced around the top and bottom edges with bands of strap iron bolted together through the flooring. *C* represents a counterbalance to assist in raising and lowering the chute. When the filled bucket reaches the collar of the shaft it is hoisted high enough to clear the chute, which, being raised, is in position *K*. As the bucket is then slowly lowered the topman simultaneously lowers the chute. A 6-in. plate of ¾-in. iron fastened to the end of a chain (about 21 in. long) suspended from the bottom of the bucket is caught in slot *H*. The bucket and chute are then slowly lowered until the top of the chute strikes the bumper *D*. As the bucket is lowered for dumping, the curved block *E*,

gouged somewhat in the center to fit the bucket and thus preventing lateral motion, raises the bottom end of the bucket high enough to allow the contents to be completely removed. After dumping, the topman raises the chute while the bucket is being swung clear, by pulling downward on the rope *G*. An important detail to be observed in building a dumping chute of this sort is to make the block *E* about 15 in. long, curved as sketched and gouged to prevent the rolling motion of the bucket. Five strips of strap iron should be securely fastened on the block for the bucket

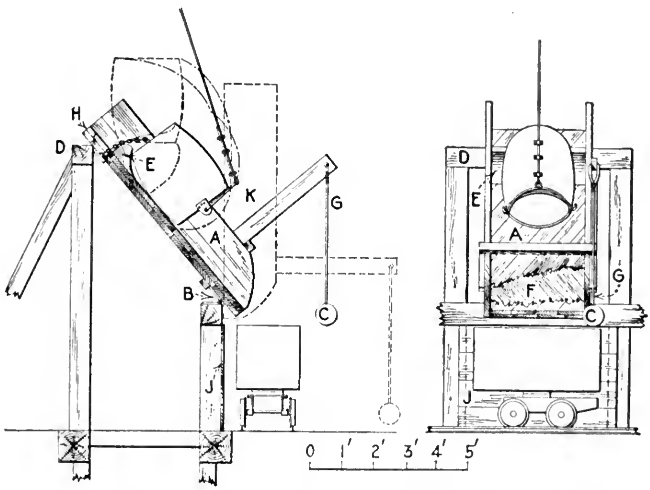


FIG. 325.—SWINGING CHUTE FOR DISCHARGING BUCKET.

bottom to slide on, facilitating dumping and affording protection against wear. The iron strips should be occasionally greased. It is advisable to board up the bent on the car side of the shaft, as shown at *J*, to prevent dirt from dropping down the shaft.

Automatic Water Bucket Dumper (By Algernon Del Mar).—When hoisting water from vertical shafts by means of a bucket, it is usual to have a man attend to the dumping at the collar of the shaft. The arrangement shown in Fig. 326, used at the shaft of the Bishop Creek Milling Co., in California, dispenses with the extra man, so that the hoisting engineer attends to the whole operation, the device being automatic. The discharge trough *D* is stationary, the portion *E* being hinged. In the position illustrated, the bucket although not shown, is supposed to be resting on *E*, and is dumping the water through a bottom valve. The engineer hoists the empty bucket, which strikes the crosshead *H*. The crosshead cannot descend below the stops *K*. As the crosshead is raised, it pulls the rope *C*, which lifts *E* to a vertical position, where it is held by the weight *F*. When the engineer hoists the full

bucket, it strikes the crosshead and pulls on the rope *A*, which is connected to the crosshead and to *E* over a pulley, again lowering the trough into the position shown; the weight *F* is not heavy enough to raise the trough alone. The ropes *A* and *B* for convenience should be on the opposite side

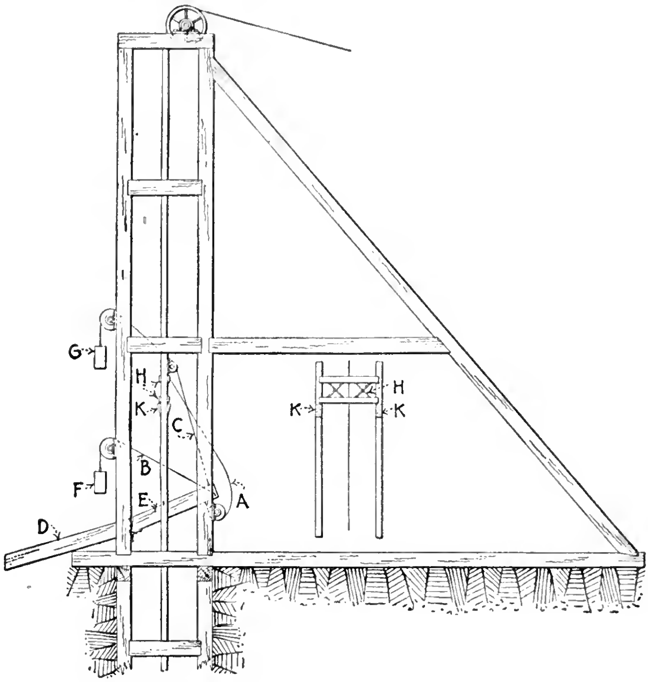


FIG. 326.—SELF DISCHARGING BAILING BUCKET.

of the trough from *C*. The rope *A* is long enough to allow *E* to be vertical without pulling on the crosshead.

Dumping Arrangement in Sinking (By L. D. Davenport).—Fig. 327 shows an arrangement for carrying a sinking bucket clear of the shaft. Two hangers of $\frac{3}{4} \times 3$-in. flat iron are bent at 90°, 6 in. from their upper ends, and are fastened by $\frac{1}{2}$-in. bolts to one of the 12×12-in. timbers under the sheave wheel. The hangers support a 1×4-in. rail, the upper corners of which are slightly chamfered to lessen the wear on the 8-in. sheave which travels over it. There are three bolt-holes in the lower end of the hangers spaced 3 in. so that the inclination of the rail may be changed; as shown, it slopes $2\frac{1}{4}$ in. to the foot. A "grab link" is suspended from the 8-in. sheave by a yoke of $\frac{1}{2} \times 3$-in. flat iron, and a short piece of chain with a hook at its lower end is caught into the grab link. A latch, made of $\frac{1}{2} \times 3$-in. stock, bent as shown, is fastened by a $\frac{3}{4}$-in. bolt to the hanger near the hoisting rope. A bumper curved to fit the sheave

is fastened to the other hanger by a short L-shaped bracket. In operation the bucket is hoisted until the hook on the chain from the grab link can be caught into the bail. The bucket is then lowered, the latch released and the small sheave carrying the bucket rolls down to the bumper. This

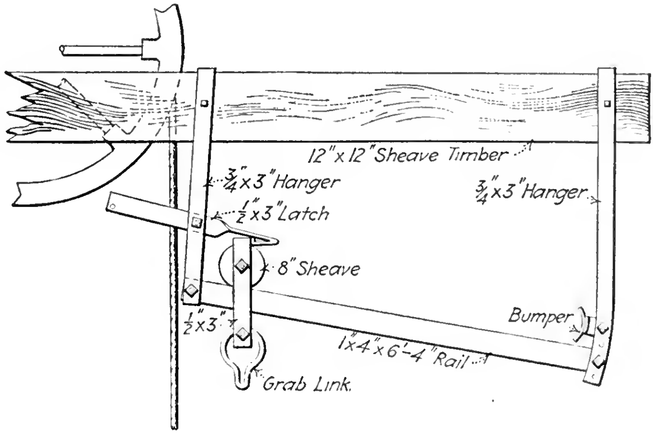


FIG. 327.—TROLLEY HOOK FOR SINKING BUCKET.

movement carries the bucket clear of the shaft and directly over a small car on the surface. The bucket is dumped, the hoist started and the traveling sheave returned to its original position, where it is held by the latch. The chain is then unhooked and the bucket lowered into the shaft.

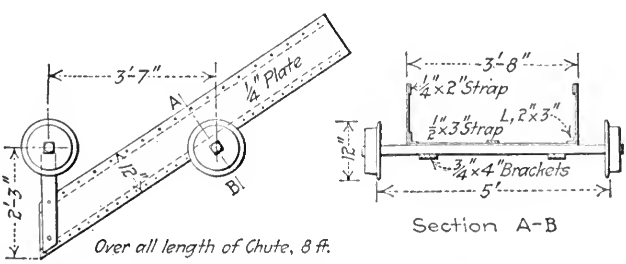


FIG. 328.—SLIDING CHUTE FOR BUCKET DUMP.

Sliding Chute for Sinking (By L. D. Davenport).—Fig. 328 shows a sliding chute in common use in the Lake Superior iron country for shaft-sinking operations. The apparatus is operated by one lander as follows: The sinking bucket is hoisted just high enough to clear the rear end of the chute and the latter is pulled back under the bucket until the wheels strike the crosshead guides. The bucket is dumped into the chute and the dirt and rock run into the tram car below. The chute is then pushed ahead clear of the shaft and the bucket is lowered. With a little practice, this

operation is done quickly. The chute is sufficiently long so that in case the bucket gets away from the lander and dumps backward, as sometimes happens, no dirt falls into the shaft. The chute is 8 ft. long over all and is built of 1/4-in. iron plates stiffened with iron straps. The side plates are fastened to the bottom plate with 2 × 3-in. angles riveted as shown. The front end is 1 ft. 9 in. below the rails which carry it and about 1 ft. above the top of the tram car at the shaft collar. This brings the chute track about 7 ft. above the collar of the shaft. Timber is brought in at the opposite side of the shaft from the tram car and since the gage of the chute track is the same as the distance between the dividers, 5 ft., there is plenty of room.

X

CARS

Types of Cars—Car Dumps—Cars for Special Purposes—Accessories

TYPES OF CARS

Doe Run Mine Car.—The car used in the mines of the Doe Run Lead Co., in southeastern Missouri, is of the type requiring dumping by a tipple. Under many conditions this is not much of a drawback, and such cars, in which the body and wheels are tied tight together, are much cheaper to keep in repair than are those pivoted to a wheel truck so as to allow dumping at any place. Moreover, they are cheaper in first cost. In order to permit placing the wheels under the body they are fastened by their axles to two 8-in., 23-lb. I-beams riveted to the bottom of the body, which is of $\frac{3}{8}$-in. plate, Fig. 329. The sides of $\frac{1}{4}$-in. plate are riveted to this bottom plate without any reinforcement at the lap. The top of the car is reinforced by a $3 \times \frac{1}{4}$-in. strap, while $4 \times \frac{1}{2}$-in. pieces are put on the inside of the ends of the car to strengthen them and to distribute the strains that come from the draw rings over a larger area of the end plates. These draw rings are used for making up trains and for mule haulage. The wheel base is 18 in. and the wheels are fitted with Whitney roller bearings. A peculiarity of the body is that it is nearly square.

Round-bottom Car.—In Fig. 330 are shown the details of the mine car used by the St. Louis Smelting & Refining Co. in southeastern Missouri. The car is made to dump by a tipple arrangement. It has a semicylindrical bottom, to which the cast-iron bearings are bolted, no truck being used. The body is made by riveting together plates $\frac{1}{4}$ in. thick. There are five of these plates; the bottom plate B', the two side plates B, and the two end plates A. The side plates and the semicylindrical bottom plates are lap-jointed, while the end plates are riveted with their flanges inside the side and bottom plates. These end plates are flanged at the boiler works where they are cut, but the rest of the building of the body is done at the company's shops. If desired, the plates could be joined together by angle irons which would stiffen the car somewhat more than the manner shown. Around the top of the car there is a stiffening band of $\frac{1}{4} \times 2\frac{1}{2}$-in. iron.

These cars have a capacity of approximately 20 cu. ft. and hold about 1 ton of lead ore, the gangue of which is a magnesian limestone. They are made to be trammed by hand or pulled in trains either by mule or by electric motor. On that account hooking rings C are provided at each end. These are U-bolts of 7/8-in. iron, put through the body, with stiffening plates of 1/4-in. iron 18 in. square put on the latest cars to spread the strain of the pull over a larger portion of the front. These U-bolts, or

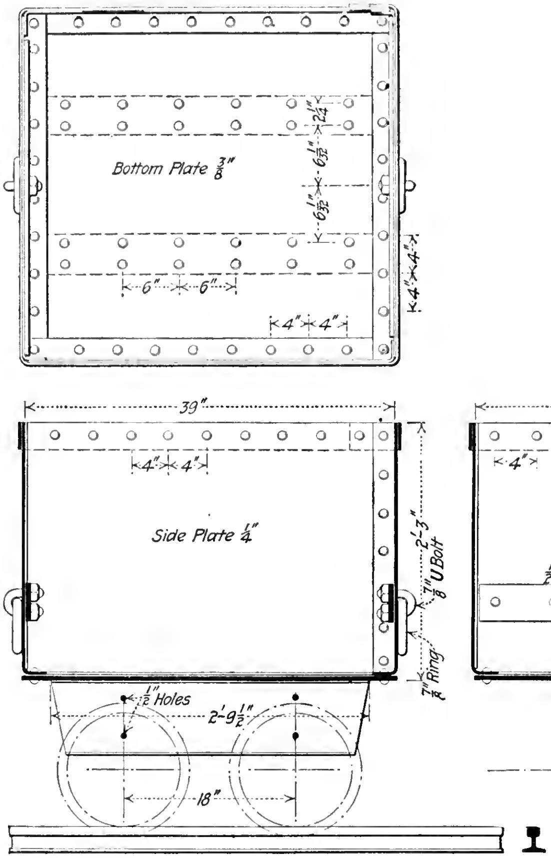


FIG. 329.—SMALL, RIGID CAR WITH SQUARE-SECTION BODY.

hooking rings, are made with collars welded on, so that they can be riveted tight to the body of the car. Cast-iron bumpers D are riveted to the body at its bottom, where the strain of the bump will be distributed directly to the reinforcement afforded to the ends by the riveting flanges. The bearings E for the wheels are connected direct to the body of the car by brackets. These bearings are of the Anaconda type, the wheels being mounted tight and the axles cut in two at the middle. The axles

are held in by the forks F. The opening in the journal is closed by a plate fastened by four cap screws. A hole is cut in it for shooting in the oil with a squirt gun, such as is used in watering holes in stopes, the intention being, however, since the oil works through the bearing rather rapidly, to use grease in the future.

Heavy End-dump Car (By H. L. Botsford).—Fig. 331 shows a tram car for underground use. It is used in connection with motor tramming, although it is equally well adapted to hand work. The car is constructed with false bottom, or lining plate, which is removable, and which can be changed end for end when the front end of the plate is worn out. The

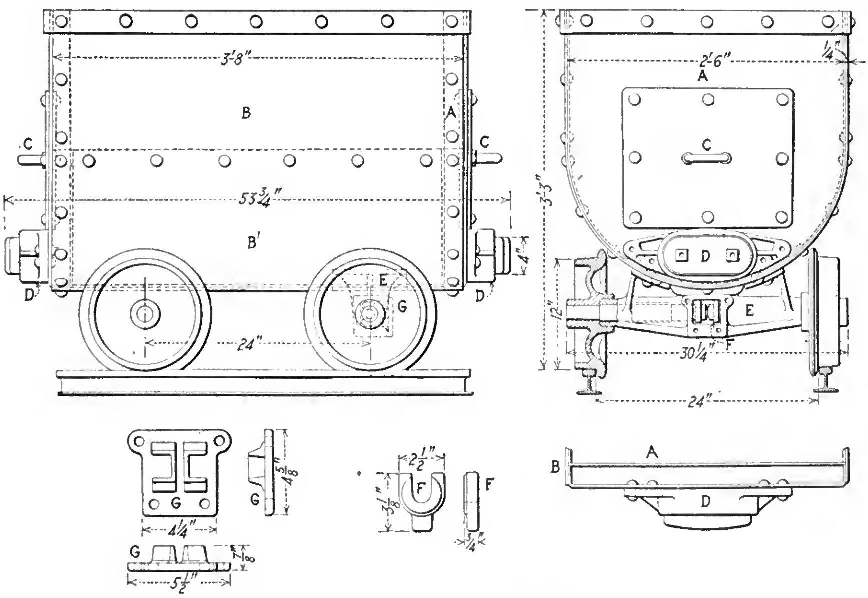


FIG. 330.—RIGID CAR WITH ROUND BOTTOM AND ANACONDA AXLES.

truck is of wood and can be readily constructed by a mine carpenter and blacksmith.

Copper Range Car (By Claude T. Rice).—The ore car used at the Baltic, Trimountain, and Champion mines, is in many respects the best mine car in the Lake Superior copper district. The body of the car, Fig. 332, is long and low so that shoveling into it is easy, and as the front end is open and the back end somewhat lower than the sides, it is much easier to fill it than some of the better known types of cars. Its capacity is 2½ tons of the Lake Superior ore, which when broken measures 20 cu. ft. per ton. In spite of the length of the car the wheel base is short, while the track gage is only 24 in. These dimensions are unique in the practice of the Lake Superior copper region. The wheels are rigidly attached to

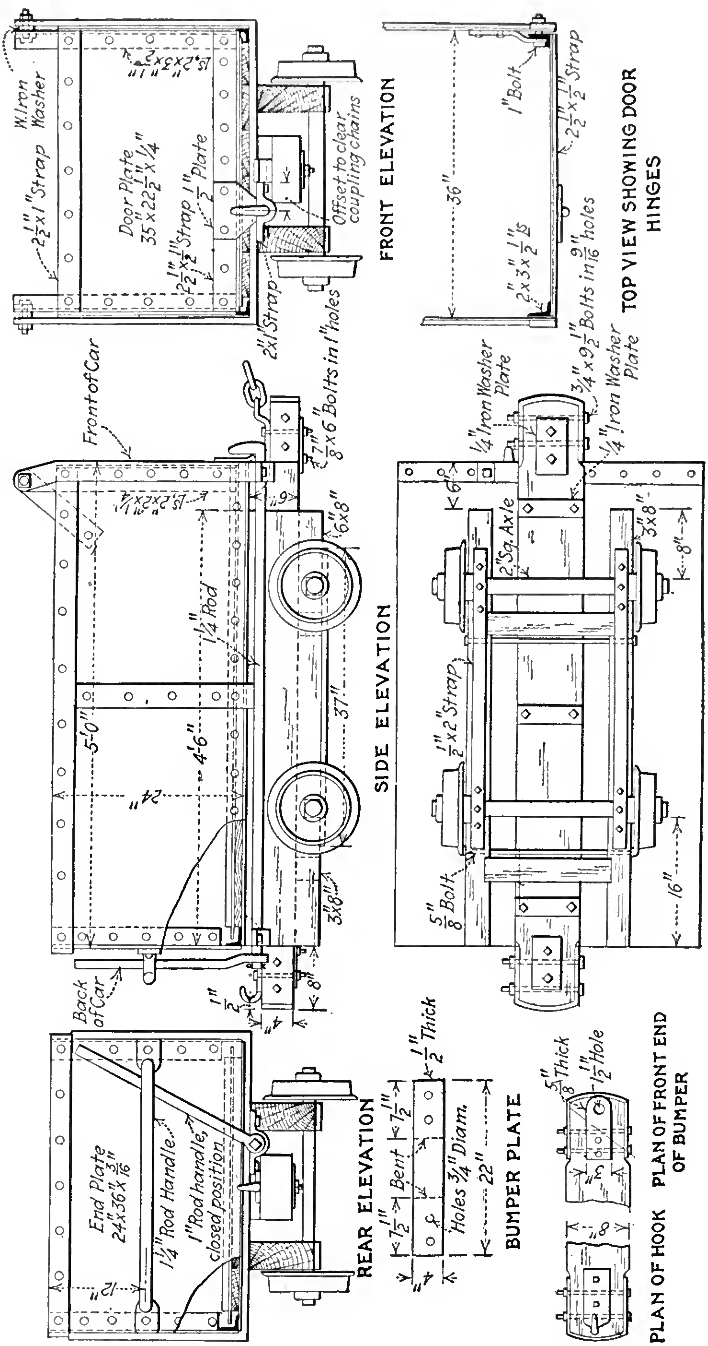


FIG. 331.—SIMPLE 30-CU. FT. CAR SUITABLE FOR MECHANICAL HAULAGE OR HAND TRAMMING.

the truck and the weight is evenly distributed upon the wheels so that the car runs easily. Bearings of the Peteler type, a modification of the regulation railroad bearings in which the journal rotates in brasses are used without waste in the oil well. The oil receptacle may be drawn out of the solid bearing casting for filling. To prevent trouble from breakage of the bearing castings, they are made of malleable iron. The bearings are carried on overhanging extensions of the axle. Because of this the axles must be made strong.

Stiffening plates, which also serve as wings by which the car is grasped when being handled on the turntables, are used on the corners of the body. As turntables are provided at all the shaft stations the cars can always be run to the breast with the front end forward and the men can shovel

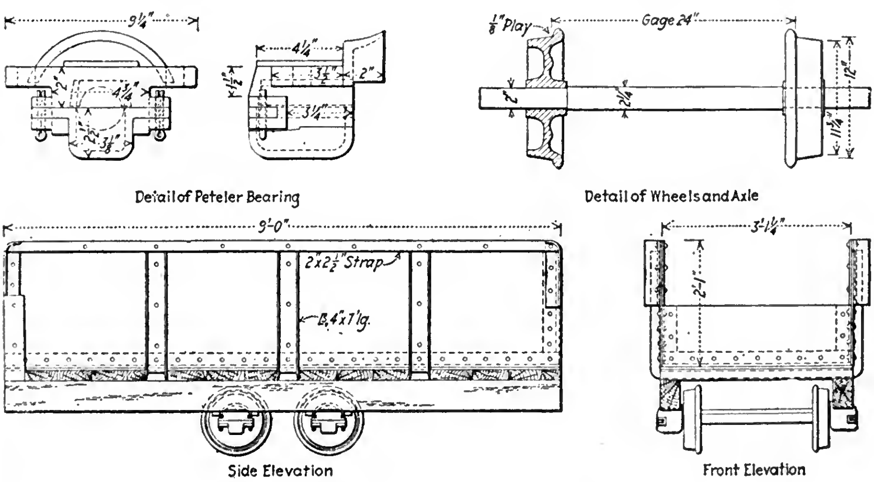


FIG. 332.—LONG, LOW CAR WITH SHORT WHEEL BASE.

into them easily. Leaving the front end open requires the building of a rough wall of boulders across the front, called by the Cornish miners a "stilling," to hold back the fine ore, but making it does not take much time. Doors were tried on the cars, but it was found that the maintenance was heavy. Because of the open end, it is possible to up-end the cars, put a brace under the bodies and roll any boulders into the front end of the car.

The truck was formerly made of hardwood, but that material rots quickly underground, and the truck is now made of fir, giving a much longer life. Practically the only maintenance on these cars is the replacing of the trucks. Owing to the type of axle, the gage of the track, the short wheel base and the even distribution of the load on the wheels, as well as the excellence of the lubrication, the tracks on the levels may be

laid at a grade of 3 in. per 100 ft. The tramming is generally done by men in pairs. Because of the length of the car it is not adapted to crooked drifts, although it goes around curves without much binding.

Desloge Car.—The mine car used by the Desloge Consolidated Lead Co., in southeastern Missouri, is shown in the accompanying illustration. The car is of the long, low type, which has found favor in the Lake copper and iron districts, due to the ease and rapidity of loading. The Desloge car stands 33 in. above the rail, is 5 ft. long, 44 in. wide and has a wheel base of 24 in. The body of the car is made up of 3/8-in. plates. It has been found that by using a car composed of several plates, as seen in Fig. 333, wrecked cars are more easily repaired than when fewer and larger members are employed. All the plates are cut to the correct size at the

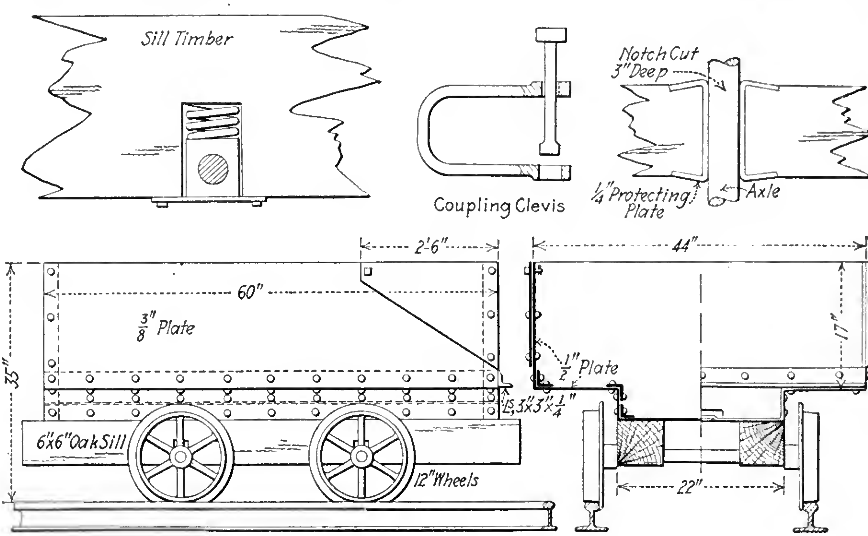


FIG. 333.—LOW, RIGID CAR WITH SPECIAL AXLES.

mill, and are assembled and riveted at the mine, 5/8-in. rivets, spaced 6 in. center to center, and 1/4 × 3 × 3-in. angles being used.

The 2-in. square axles have collars immediately behind the wheels, which prevent any side play. An unusual feature of the car is that these axles can move up and down in iron-lined notches cut in the truck timbers, 2 in. wide and 3 in. deep. This enables the wheels to follow any vertical inequalities of the track. In order to make the car itself ride more easily, coil springs are fitted in the notches between the axles and the truck timbers. The longitudinal truck timbers project 3 in. at both ends and act as bumpers. The car body being fastened rigidly to the trucks, tipples are used in dumping; the cars have squirrel-cage doors which permit the use of the spear tipple. The link-and-two-clevis coupling arrange-

ment is generally used. In order to prevent pins from being lost, the upper hole of the clevis is made smaller than the lower one, the pin is put through the upper hole and upset just enough to prevent it from being pulled back, while being still small enough to go through the lower hole easily.

Two Sublevel Cars (By H. L. Botsford).—In the sublevel system of mining on the iron ranges there is need for a special tram car which can be easily brought into and removed from the subdrifts. The only access to the subdrifts is usually by a raise or winze through which it is impossible or inconvenient to take the main-level cars.

Figs. 334 and 335 show two types of small tram cars which are well adapted to this work. They are so constructed as to be readily dismantled for moving and quickly assembled again. A light steel car is shown

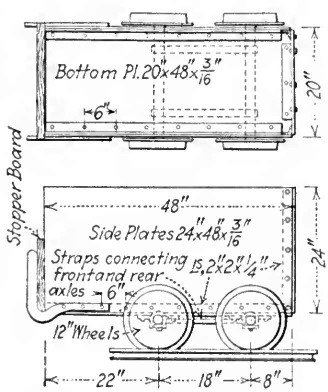


FIG. 334.—STEEL SUBLEVEL CAR.

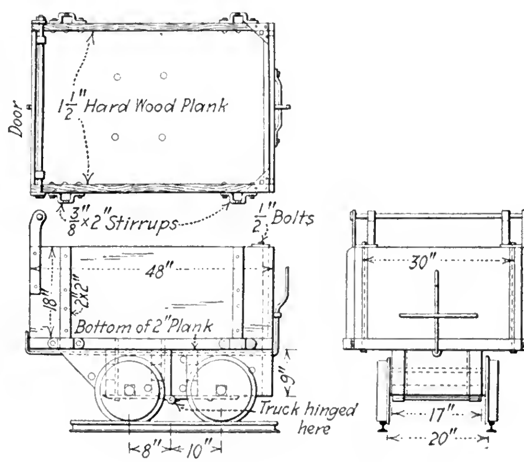


FIG. 335.—WOODEN SUBLEVEL CAR.

in Fig. 334. The sides are bolted to the bottom and the end to the bottom and sides. The front axle only is bolted to the car box. The front and rear axles are connected by straps, so that the box lifts off the rear axle when being dumped. A wooden car, built with detachable sides and end, is shown in Fig. 335. The truck is hinged, and only the front half is fastened to the car box. The sides of the car are provided with cleats which fit into stirrups bolted to the bottom planking.

Pickands-Mather Sublevel Car.—The accompanying drawing, Fig. 336, represents the standard car designed for sublevel work in the Pickands, Mather & Co. properties on the Mesabi range. The nature of the work is such that cars must be of a size to permit easy handling in narrow, crushed drifts and around sharp curves. The car shown has a capacity of 20 cu. ft. which is equivalent to about a ton of loose ore. This

is an advantage in keeping a record of the work performed by various gangs. The truck is made strong to withstand rough usage and the body can be easily straightened if accidentally dropped down a chute. The tipping device is placed near the center of the car, which makes it easy to dump. The length of the car, 5 ft., in proportion to its wheel base, is a noteworthy feature. It is also comparatively low, for its breadth and length, which renders shoveling easier. The car will dump only toward the front. The 12-in. wheels turn on the axles. The axles are rigidly attached to the sills, which are made of channels and braced by plates over the top and on one end.

Federal Gable-bottom Car.—Fig. 337 shows the details of the gable-bottom car used on the electric haulageways at the No. 1 shaft of the Federal Lead Co. in southeastern Missouri. This car has a capacity of 28 cu. ft. The locking mechanism is worthy of mention, as it is rapid and easy to operate. It consists of the shaft F to which the double lever D and

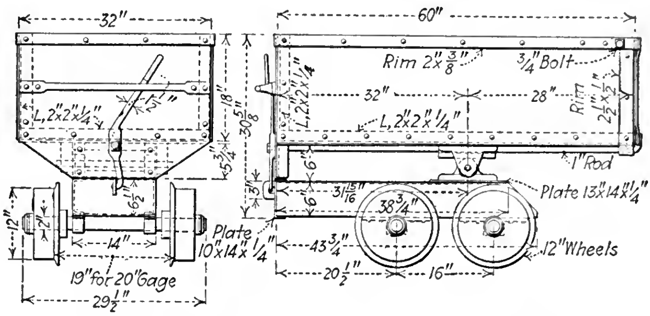


FIG. 336.—FRONT DUMPING CAR USED FOR SUBLEVEL WORK.

the operating lever A are keyed. To the ends of the double lever D are pinned the door levers B and C, which are bent to let their pin connections with lever D come into dead-center positions with respect to the pins E when the doors are shut, thus locking the doors. The outer ends of each of the door levers B and C go over the pins E, which are bolted to the doors of the car. Each end of the car is fitted with the locking mechanism, but as levers D of both ends are keyed to the same shaft F, the unlocking of one end of the car also unlocks the other end. The bar used in removing any boulders that hang up in the car may be used to trip the locking mechanism by catching the link-bolt of one of the locking arms and giving it an upward lift. This is more convenient and easier than the use of the locking lever A for opening and closing the doors. In fact, the lever A may be entirely omitted if desired. The locking mechanism is made of cast steel.

Double-truck Gable-bottom Car.—Fig. 338 illustrates a car designed for use on the stockpile trestle of the Kennedy mine on the north Cuyuna

iron range. The car will hold 100 cu. ft., or about 6 tons of ore. The electric locomotive in use on the trestle will handle one car per trip. The body of the car is hopper-shaped with a gable through the center and two

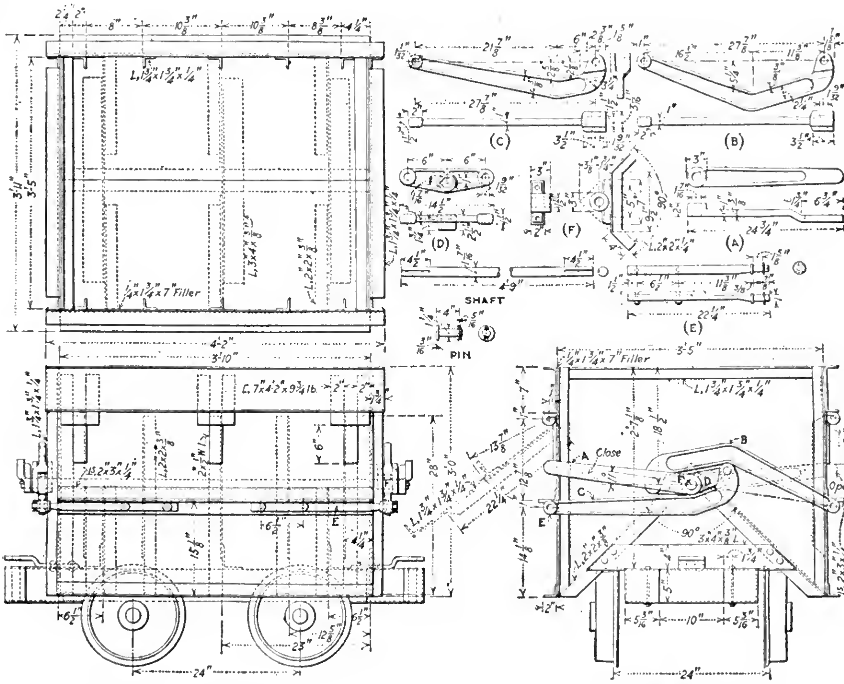


FIG. 337.—DETAILS OF FEDERAL LEAD CO.S GABLE-BOTTOM MINE CAR.

side doors swinging out and up. The ends of the hopper are pitched at 60° and the gable is about 50°, these steep pitches being necessary from the fact that the ore is sticky and hangs to a flat slope. It is possible also

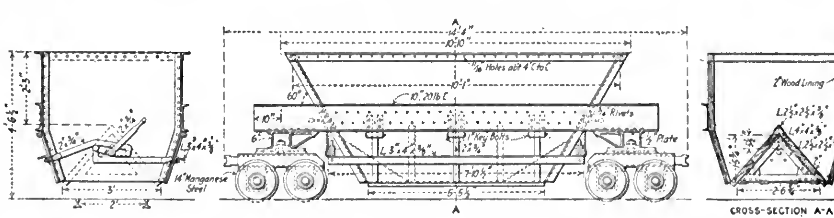


FIG. 338.—RAPID-DUMPING 6-TON CAR FOR STOCKPILE TRESTLE.

that in the coldest weather the ore may freeze slightly, which would cause it to stick to the car. The inside of the body is protected with a 2-in. lining of wood. The two side gates give an extremely fast discharge, since they provide about a maximum opening for the passage of the ore.

It is extremely important that the cars dump quickly, as it will tax the capacity of the system to handle the ore as fast as it is hoisted at best, the trestle being unusually long considering the size of the mine. The double trucks make it possible to swing the entire body in between and thus keep it lower and more stable. The company itself manufactures everything but the trucks.

Middle-dump Mine Car (By W. W. Shelby).—The 2-ton middle-dump mine car, shown in Fig. 339, has been in service for years at the Smuggler-Union mine. The sides and bottom are ¼-in. iron and are held together by 2 × 2 × ⅜-in. angle irons. A 1½ × ½-in. strap is riveted around the top of the car. The chain across the top at the middle prevents deformation and makes the pivots, about which the car dumps, more rigid. Three-inch planks are used for lining the bottom and for bumpers, the rest of the car being entirely of iron. In order to dump the car, lever *A* is

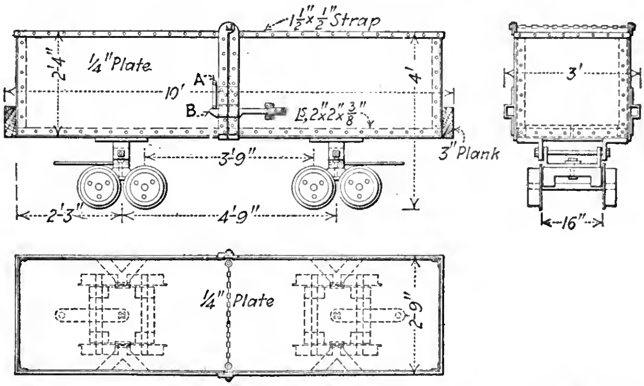


FIG. 339.—SMUGGLER-UNION CAR THAT BREAKS IN THE MIDDLE.

knocked to one side and latch *B* forced upward by a light blow from a pick or a single jack; the car will then break in the middle, dropping the ore into pockets or bins beneath the track. The trucks are so spaced that the load between them is only a little greater than that in the ends, thus avoiding undue strain upon the latch and its safety lever. The truck has fixed wheels and is unusual in that there is no bottom half of the journal, the lower half of the axle being exposed. A U-bolt extending beneath the axle holds the wheels to the truck when the car is lifted from the track. Those cars that come to the surface are automatically oiled by passing over two rollers of such dimensions and so placed between the rails, that the lower parts of the axles beneath the bearings touch them in passing. These wooden oiling rollers are free to revolve; a sack or cloth is tacked on their circumferences, and being suspended in a tank of oil, they are themselves automatically oiled and furnish a good lubricating surface to the axles. The cars are coupled by chains.

Red Jacket Car.—The car shown in Fig. 340 is used at the Red Jacket shaft of the Calumet & Hecla company in the copper district of Michigan. The most interesting feature of the car is the manner of reinforcing the bottom. Two plates are used, bolted together to form a beam, with a filling of wood to give stiffness. The top plate is $\frac{3}{8}$ in. thick while the bottom plate is $\frac{1}{4}$ in. thick, with wood filling 2 in. thick at the center. The gage of the track is 48 in. and the car has a capacity of $2\frac{1}{2}$ tons of conglomerate ore. This car, owing to the construction of the bottom, carries the load without strain, although the wheels are carried by trunnions fastened to the sides of the car instead of by axles extending clear

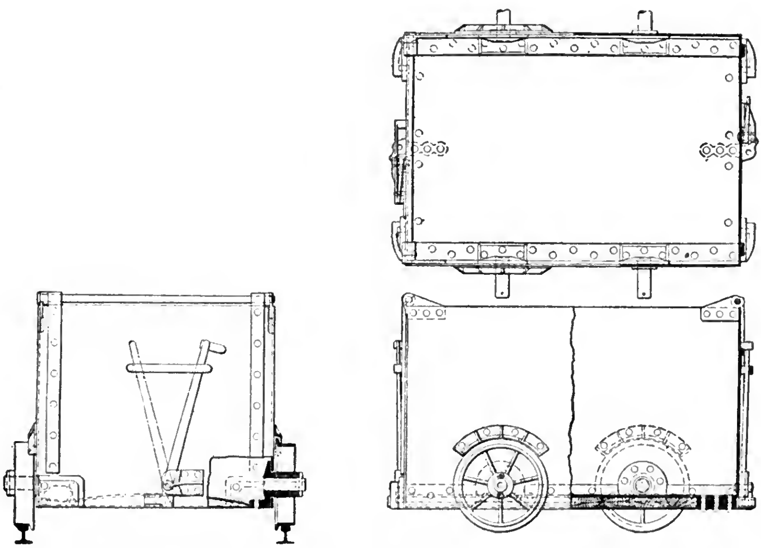


FIG. 340.—CALUMET & HECLA CAR WITH WHEELS ON TRUNNIONS.

under the body. This is partly because of the limits on size imposed by the size of the cages in the Red Jacket shaft, and partly because the body of the car had to be kept as low as possible, for use with rope haulage. Axles could not be used and have the cars at the desired capacity. The car is fitted with doors at each end to aid in loading boulders from the floors of the drifts when cutting-out stoping is being done, although most of the ore after stoping proper begins is loaded from chutes. As the car is used on the rope haulage levels, buffers are provided at the corners. The car is dumped by a chain coming down from an air cylinder over the measuring pockets at the shaft. These cars stand up to the work well and no trouble comes from bowing of the bottom. The same design of bottom is used for the $7\frac{1}{2}$-ton cars serving the underground bins at Osceola No.

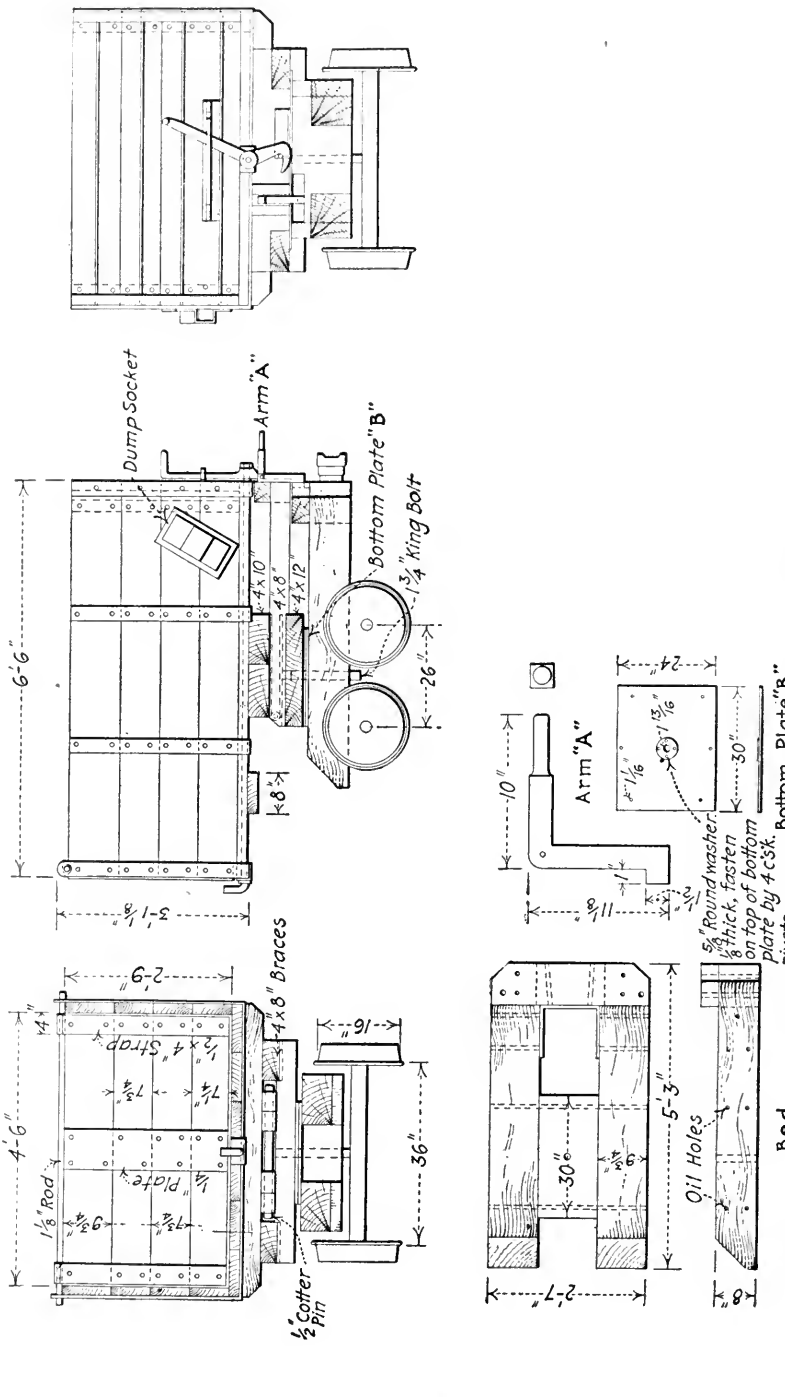


FIG. 341.—OAK STOCKPILE CAR FOR MOTOR HAULAGE.

13 shaft; here the track has a gage of 5 ft. 9 in. in order to keep the car low and short for dumping.

End-dumping Stockpile Car of Wood (By E. W. R. Butcher).—Fig. 341 shows a stockpile car of the front-dump, turntable type, used by the Republic Iron & Steel Co. at some of its mines on the Mesabi range. The size of the car varies with the size of skip used. It is handled by an electric motor to which it is attached with an iron bar $1\frac{1}{2} \times 4$ in. by 8 ft. This bar acts as a protection to the motor when the car goes over the end of the stockpile. The car is built largely of wood; all of this is oak, which, although much more expensive than pine, needs less repair and is cheaper in the end. The bottom of the car is lined with a $\frac{1}{4}$-in. iron plate. Formerly the bottom lining consisted of 1-in. plank, but this had to be replaced too frequently. The release handle, operating in about the usual manner, allows the car to dump on its hinge and opens the door at the same time. Should the car not dump of itself, a wooden lever arm is placed in the

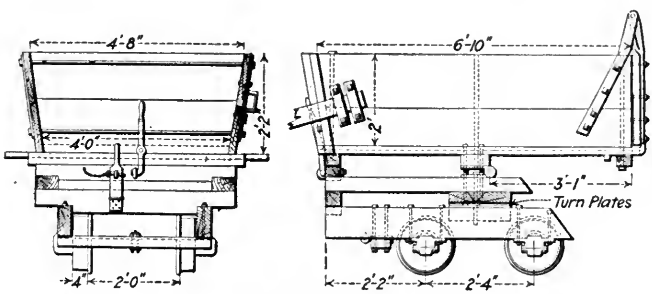


FIG. 342.—WOODEN, END-DUMPING TURNTABLE CAR.

dump socket on the side and the car raised until it does dump. The 4×8-in. timbers on either side of the hinge act simply to steady the car. The king bolt for the turntable is not fastened to the truck; this allows it to pull out and release the body of the car from the truck when it goes over the edge of the stockpile. It has been found less trouble to get back the body of the car after it has gone entirely over the pile, than to get the truck and car back when they are hanging on the edge. To release the car for swinging, arm A is raised; until it is thus raised, a $1 \times 1\frac{1}{2}$-in. extension at its bottom, fitting under a block, prevents the car from rising, while this same extension, together with the vertical portion of the arm, is held so as to prevent any side motion. When the car is swung back again to normal position, the arm falls by itself into its place. The four corner bolts which hold the bottom plate B to the bed, also hold the journal boxes, thus eliminating four bolts. The 5-in. iron washer on this plate gives a smaller friction resistance to turning and can be easily replaced when worn.

Surface Tram Car.—Fig. 342 shows a not uncommon type of tram car

around the iron ranges for use on the surface to handle skip-hoisted material. It is of the end-dump, turntable type and is constructed chiefly of wood; it is rather lower than those frequently used. The socket at the rear end is for the insertion of the dumping bar. The manner of suspending the door some distance above the top of the car is noteworthy.

CAR DUMPS

Rack-and-pinion Front-rotating Dump.—In Fig. 343 are shown the details of construction of the dump or tipple used at the mines of the St. Louis Smelting & Refining Co. in southeastern Missouri, for dumping cars made with the truck fastened rigidly to the body. In this tipple the cradle proper carrying the car wheels rides ahead on a framework

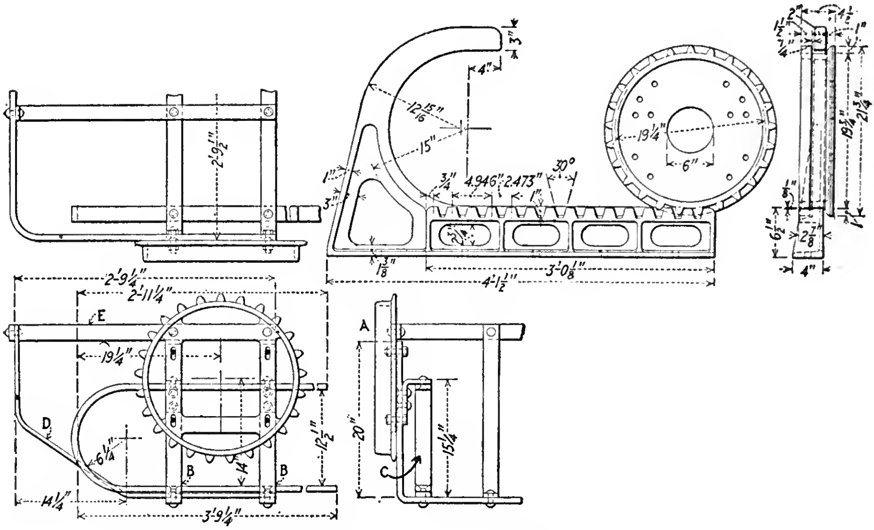


FIG. 343.—CRADLE DUMP OF ST. LOUIS SMELTING & REFINING CO.

before it dumps the car. The frame pieces on which the cradle travels end in hooks or horns that catch the cradle and keep the car from going clear over; consequently, no reliance has to be placed upon a chain fastened to a crossbar of the cradle and anchored to a tie of the track or fastened in some other way to solid ground. This type of dump is somewhat more expensive than the type in which the cradle swings on its axle and does not travel forward.

The frame on which the cradle travels is made up of two castings bolted to timbers running out across the chute into which the ore is to be dumped. The castings, as stated, have horns that come up and hook over the wheels of the cradle and since the cradle wheels have teeth which

mesh with teeth on the frame castings, the cradle can turn up only a certain distance. These teeth do not carry the weight of the cradle, for the casting extends up to form a track ledge slightly higher than the tops of the teeth on which the wheels of the cradle roll. The function of the teeth is only to regulate the forward motion of the cradle wheels and keep them even as well as to limit the end of the dump when the wheels strike the hooks of the frame pieces. The cradle dump wheels *A* are fastened to the cradle frame by bolts that pass through slots in the crossarms of the wheels. In this way it is possible to level the cradle frame with respect to the dump wheels. The cradle frame is composed of the two cross-braces *B* to which the dump wheels are bolted and to which the track straps, and the loops *C*, for catching the car wheels, are riveted. A center strap *D* is fastened to the bridle strap *E*, which extends forward to catch the front of the car and take part of the strain that would otherwise come down to the loops *C*, while the car is dumping. In case the cars will not right themselves after they have dumped, it is customary to fasten weights to the back cross-strap *B*. The track straps *C*, of course, extend back farther than the rest of the dump so as to join with the rails of the track leading up to the tipple. When the car has righted itself and the cradle has run back to its proper position on the frame, these drop down upon a crosspiece of iron so that they come just level with the tops of the rails of the approach.

The dump illustrated is designed for a 1-ton car of the usual type used in hand tramming at mines. The drawing gives the manner of proportioning the different members of the cradle and frame so that they will stand up to the strain of the work, but the dimensions of the cradle have to be changed, of course, to suit the dimensions of the individual car that they are to serve. The dump is made of cast steel; it has been in use for several years and works satisfactorily. Some trouble has been experienced owing to the occasional breaking of a hook leg when a car has been run into the dump with more than ordinary force.

Simple Cradle Dump (By Claude T. Rice).—In the accompanying illustration, Fig. 344, are shown the details of the car dump used at the Doe Run, the St. Joseph, the Federal and some of the other mines in the Flat River district of Missouri, where the ore is hoisted to the surface in cars. This tipple is made entirely of iron, and as shown in the drawing, is designed for dumping a 1-ton car. The iron bars that serve as rails run back to a sole plate under the track rails leading up to the dump so that the bars will come flush with the tops of the rails. The other ends of these bars are bent and brought back far enough to come over the tops of the back wheels when the car has been run forward on the dump. To strengthen these loops they are tied to the lower part of the cradle by two straps, offset to the outside. The dump is so proportioned that the

center of gravity of the loaded car will be slightly ahead of the rotating shaft of the dump, when the car wheels are engaged by the loops, yet not far enough ahead but that, after the ore has run out, the weight of the back rails of the dump will be enough to put the car and tipple back into a horizontal position. Consequently the proportions have to be made according to the dimensions of the cars used; the drawing shows only the general design of a style of tipple that has given excellent service after years of use. To prevent the tipple or dump from turning so far that it cannot right itself, a chain is tied to the back and is fastened to the floor above the bin into which the ore is being dumped.

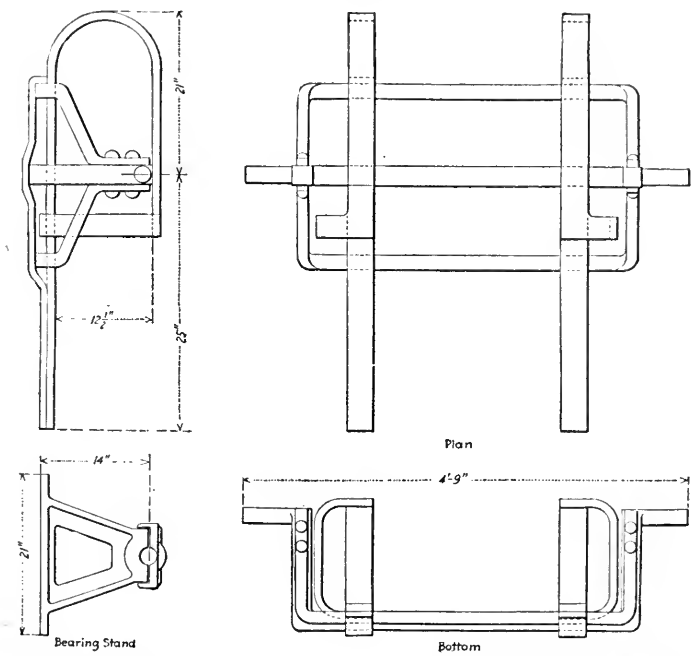


FIG. 344.—TYPICAL DUMPING CRADLE USED IN SOUTHEASTERN MISSOURI.

Tipple for Seven-car Train (By J. R. McFarland).—At the Cactus mine of the South Utah Mines & Smelters, a dump was operated which unloaded seven 4-ton cars at a time. The dump consisted of a steel cylindrical framework of a length equal to seven cars, which rested on four pairs of rollers, as shown in Fig. 345. Rails on which the cars ran in were laid in the bottom. Two angle irons extended the full length on the inside of the top in such a position as to embrace the tops of the cars, as can be seen in the illustration. An air-driven engine placed opposite the side of the cylinder, drove a drum carrying a wire rope which was also wrapped around the cylinder. The motor brought a train of 14 cars

from the mine and backed seven cars into the dump, the engine was started and the cylinder rolled over to the position shown in the lower drawing, with the inverted cars resting on the angle irons inside the frame. It was then rolled back, righting the cars, which were run out by the motor, and the other seven cars run in and dumped. This arrangement, which proved itself a valuable timesaver, was built by Silver Bros., Salt Lake City, Utah.

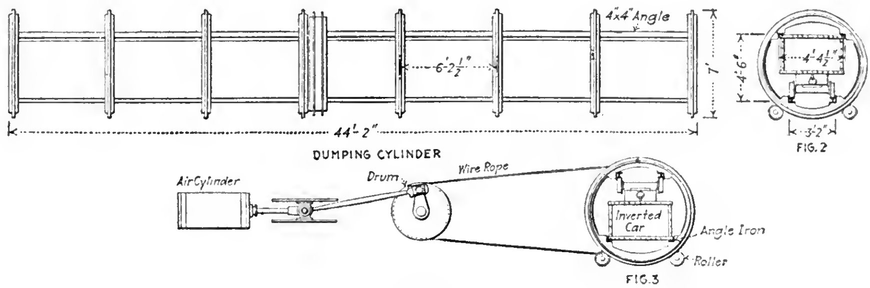


FIG. 345.—CYLINDER FOR DUMPING A TRAIN OF SEVEN CARS.

Tipple for 3-ton Car.—The complete turnover tipple, shown in Fig. 346, was designed to unload the 3-ton end-closed cars used in the magnetite mines at Mineville, N. Y. The position of the tipple axles depends upon the style of car to be dumped. The train of cars is uncoupled some distance from the tipple and dropped by gravity to the tipple, where they turn through an angle of almost 180° to dump. The cars are righted by applying pressure to the hand brake, until the projecting rails rest on the

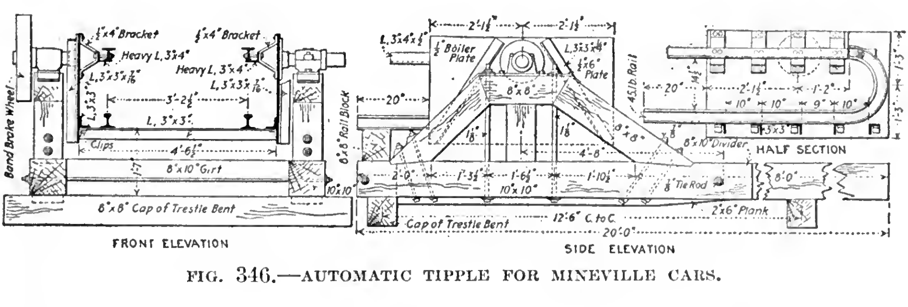


FIG. 346.—AUTOMATIC TIPPLE FOR MINEVILLE CARS.

8 × 8-in. rail block. The cars after dumping are started from the tipple by hand and the grade from the tipple takes the cars to the "empty" spur by gravity.

Spear-type Dump.—In Fig. 347 are shown the details of the construction of the spear type of car dump which is used in many coal mines, but which is not so frequently seen at metal mines. It is used, however, by the Desloge Consolidated Lead Co. in southeastern Missouri. This

type of dump has its advantage, especially when low, long cars are used. Any car that is to be dumped with a spear type of tipple must be fitted with a squirrel-cage type door, and this door must have a loop extending above it for catching on the spear, so that it will be raised in respect to the body of the car as the tipple dumps the load. The long car body has a good deal of side movement from variations in the way that the car is seized with respect to the rails; therefore the loop for the spear should be at least 10 in. wide, and 8 in. high.

In the spear type of tipple the car body is dropped enough by the cradle so that the bottom of the car assumes an angle greater than the angle of repose of the material. The front of the body then strikes a

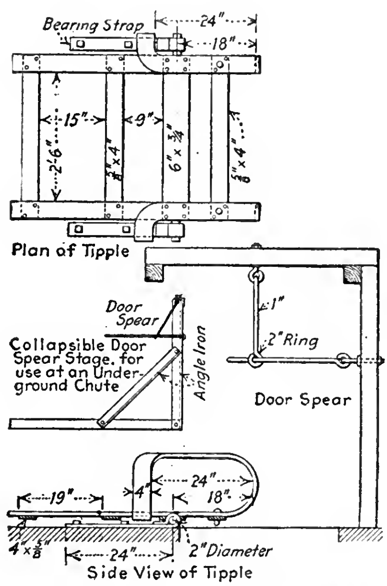


FIG. 347.—DESLOGE DUMP AND ARRANGEMENTS OF SPEAR.

crosspiece or the floor, so that it cannot tip farther. In rigging up a dump at a chute where the roof is so high that the spear cannot be suspended from it, two uprights are carried up from the stringers which support the cradle bearings. These uprights are fastened by a single bolt in each side; on these two bolts the uprights can turn as on pivots. Angle braces, also fastened by a single bolt at each end, secure the uprights from pushing forward when the door strikes the spear. When blasting near-by, the bolts of the strap braces can be taken out and the whole frame lowered down on the ground where it will not be damaged by flying rocks. The front support of the spear must be arranged so that it will slide back on the spear and lift up the free end, as the car comes for-

ward into the dump. The spear, furthermore, must catch the loop on the car door a little ahead of the tipping of the car, so that, if for any reason there is a failure of the spear to catch the loop, the car will not tip with the door unlifted, possibly causing trouble through overbalancing of the car.

Sublevel Car Dump (By L. D. Davenport).—Various kinds of tipples for end-dump cars have been tried in the Chisholm, Minn., mines at different times. Most of these required frequent repairing, and trouble was caused by fine ore getting under the rockers. Cars hinged so that they would dump without the wheels leaving the track were also used at one time. One of the objections to these cars was the inconvenience in taking them from one level to another. At present all sublevel cars are made with the boxes fastened directly to the trucks. These cars are of 25- and 41-cu. ft. capacity with 14-in. wheels running loose on the axles.

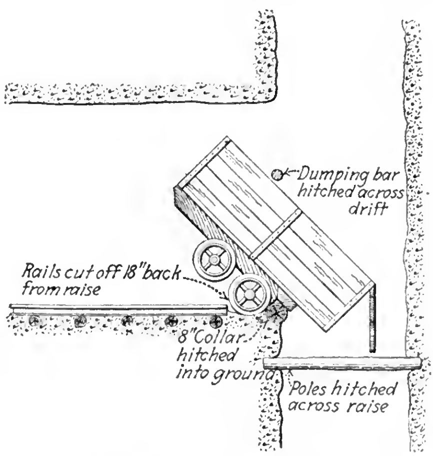


FIG. 348.—SIMPLE FIXED DUMPING ARRANGEMENT.

Fig. 348 shows a typical car dump as made at present. The rails are cut off 18 in. back from the raise and a collar-piece of round timber is hitched into the ground at the edge of the raise. A dumping bar is placed 4 ft. 6 in. above this collar and parallel to it. The loaded car comes down the track and just before reaching the dump the miner throws the lever which unfastens the car door, the front wheels drop about 4½ in., the front end of the truck strikes the collar-piece and the car dumps at an angle of about 45°. The dumping bar prevents any chance of the car's going into the raise. This bar is sometimes made of a short piece of rail. Poles or short pieces of rail are hitched across the raise to prevent anyone from falling through. These poles are placed low enough to allow plenty of room for the car door to swing open. After the car has dumped, one or two miners grasp the handle at the rear end and with one pull the rear wheels come down on the track, the front wheels are lifted clear of the

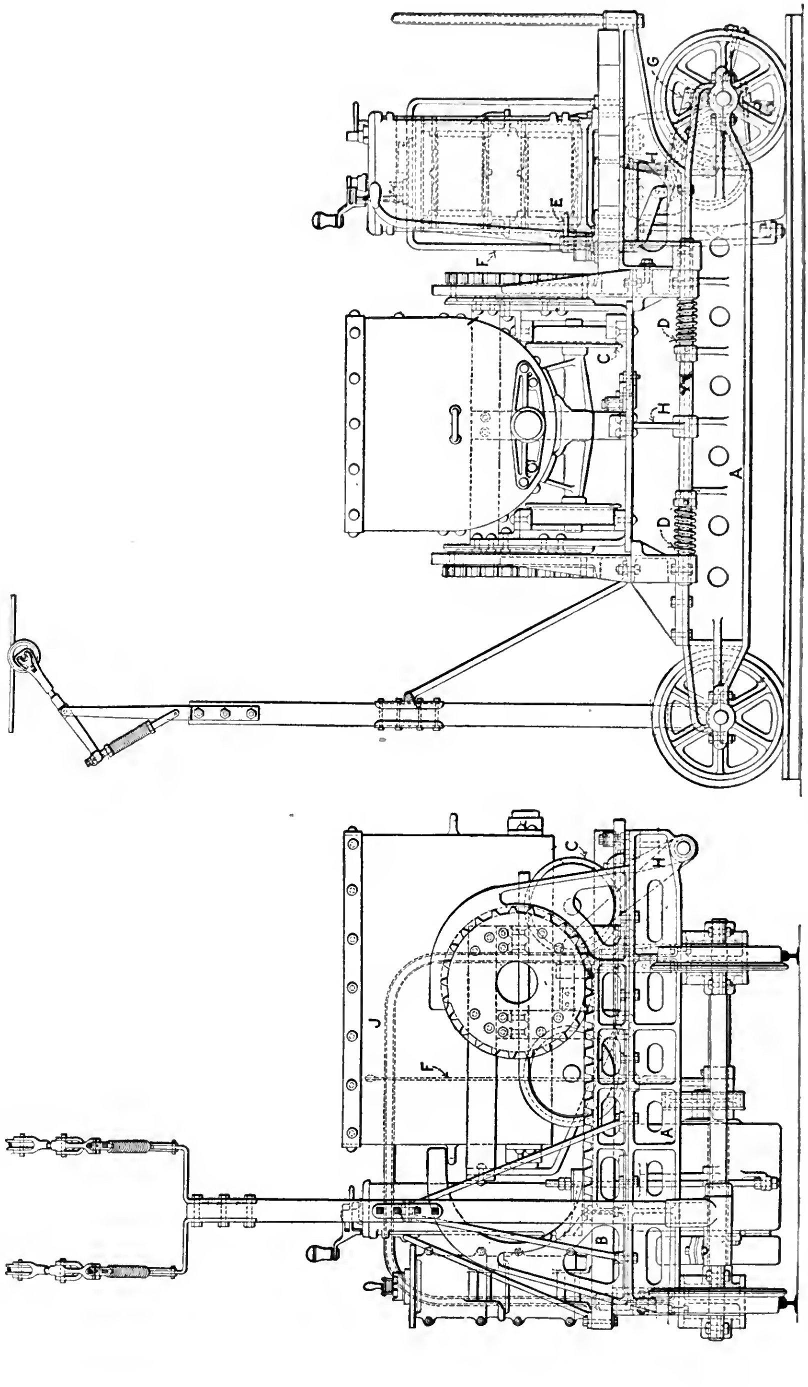


FIG. 349.—TRANSFER CAR USED IN SOUTHEASTERN MISSOURI.

ground and as the car moves back at the same time, the front wheels also come down on the rail. This dumping arrangement for end-dump cars is simple, safe, easily made and requires little repairing.

CARS FOR SPECIAL PURPOSES

Combined Transfer and Dump Car.—In the accompanying drawing, Fig. 349, is shown the construction of the transfer cars used at the shafts of the St. Louis Smelting & Refining Co. in southeastern Missouri. The ore is hoisted to the surface in 1-ton cars on single-deck cages, and at the surface, instead of tramming the ore cars to the bins by hand, they are put on transfer cars which are equipped with motor drive. The method has its advantages and its disadvantages, but the construction of the transfer car is interesting as the idea is applicable to transfer cars used for other purposes. The frame *A* of the car is of cast steel. Longitudinal members extend out beyond the deck of the car far enough to take the bearings for the wheels. The 14-hp. motor is mounted to drive the back axle, and the braking is done on the wheels of that axle. The controller is carried on a raised platform resting on a cast-iron frame bolted to the main frame of the car. To the main frame and running crosswise with it is bolted the dump frame *B*. This dump is of the rack-and-pinion or traveling-cradle type which is especially good in this class of work.

The mine car is locked in place by a locking dog so that it cannot run back off the cradle. This is shoved in place by the lander's foot when he puts the mine car on the transfer car. The accidental dumping of the cradle is prevented by the hook *H*, which is held up in place against the cradle by the hook springs *D*, as shown in the drawing. The hook is keyed to the hook shaft, as is the operating lever arm *E* by which the hook is pulled back off the crossbar of the dump cradle at the proper time. This lever controlling the dump is operated by the motorman's foot. The lever *F* extending up to the height of the controller is the brake lever which is operated by hand, thus allowing the brake to be put on under closer control than if a foot brake were used. It is by this brake that the car is stopped even with the landing tracks. The brake link-lever *G*, is fitted with a spring *I*, which holds the brake beam off the wheels. To prevent the motorman from falling off the car, a railing of pipe *J* is carried around the controller platform. A return wire is used on the trolley circuit so that a double-runner trolley pole is needed.

Car-transfer System in Rockhouse.—Figs. 350 to 353 illustrate a rockhouse used for ore crushing and storage. The ore is hoisted in cars on cages instead of in skips, and auxiliary cars are used to carry the underground cars to the grizzlies, dump them and return them to the cage. Fig. 350 shows a plan and section of the working floor, or feeding

floor for the crushers, together with the caging or transfer floor around the shaft. The loaded car is pushed from the cage, trammed a few feet, and loaded on the transfer car shown in Fig. 351. This is in effect a movable tipple. It operates on a track laid in a pit. The underground car mounted on this transfer car is trammed to a point opposite the grizzly, dumped over the latter and trammed further to a track leading at right angles past the shaft. Here it is pushed from the tipple transfer car and trammed on its own wheels to another transfer car, which also operates

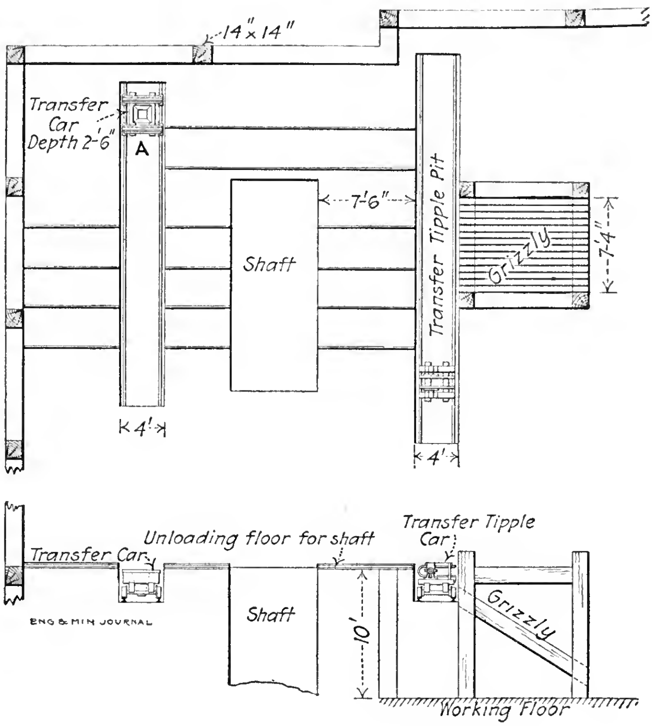


FIG. 350.—LAYOUT OF CRUSHER HOUSE.

in a pit on the other side of the shaft. Loaded on this, it is trammed to a track leading into the shaft on the side opposite to that from which it was unloaded and is then ready for loading on the descending cage. Fig. 352 shows this second transfer car. Its peculiar design is for the purpose of taking care of tank cars, in which water is hoisted from the mine, and providing means of emptying the water into a discharge launder. The underground car itself is shown in Fig. 353. This method of hoisting, of course, is far inferior to the use of skips in point of economy. Its use in this case seems to be a survival of old practice.

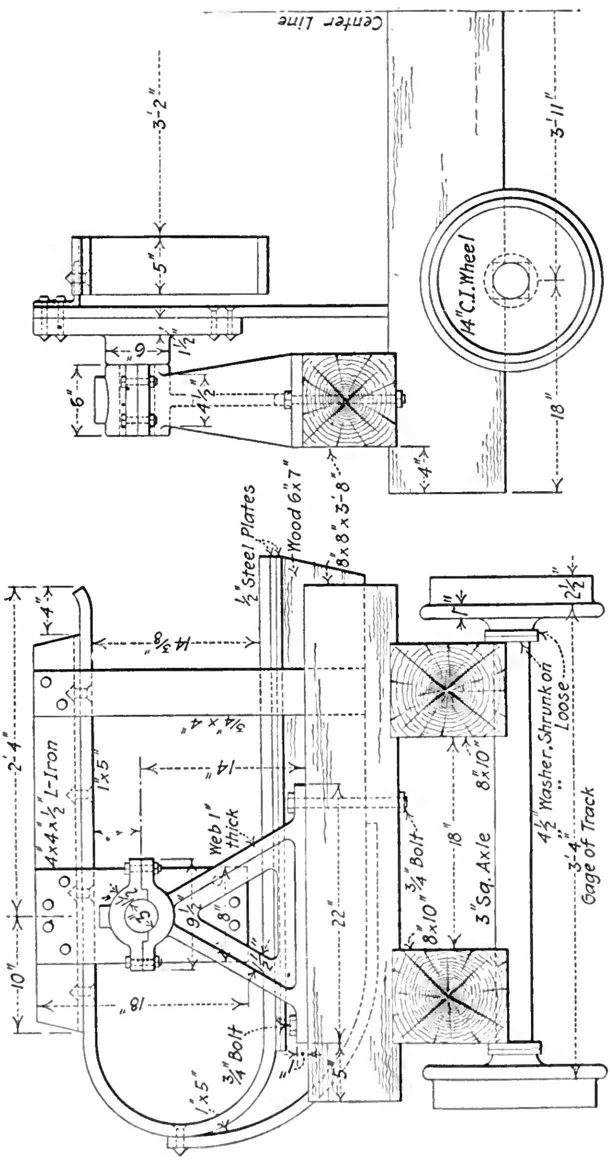


FIG. 351.—DETAILS OF TRANSFER TIPPLE CAR.

Car for Tramming Drill Steel.—In the Joplin district the ore is trammed almost entirely in buckets. These buckets rest upon low, flat-topped cars that are not convenient for transporting steel, dynamite, machines and other supplies about the mine. In Fig. 354 is shown a suitable car for such purposes. The body of the car is raised high enough from the track to be convenient. On top of this is a semicircular rack

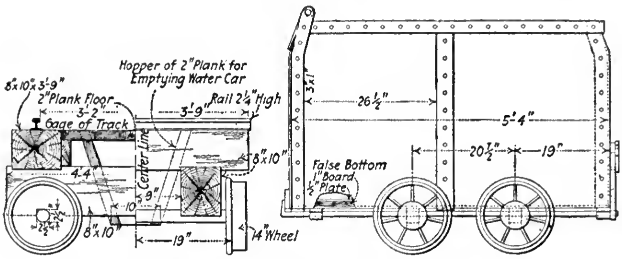


FIG. 352.—TRANSFER WATER CAR.

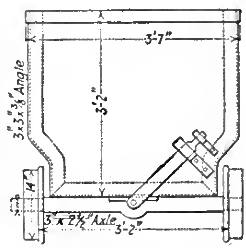


FIG. 353.—ORDINARY MINE CAR.

made of 2 × ½-in. strap iron. It is strapped together longitudinally only at the top so that in tramming dynamite to the underground magazine the boxes can be piled flat on the deck of the body, which is about 12 in. wide. The other details of the car are evident from the drawing.

Skip Car for Flat Grades.—The details of construction of a skip car for flat grades are shown in Fig. 355. These cars are used in the Joplin

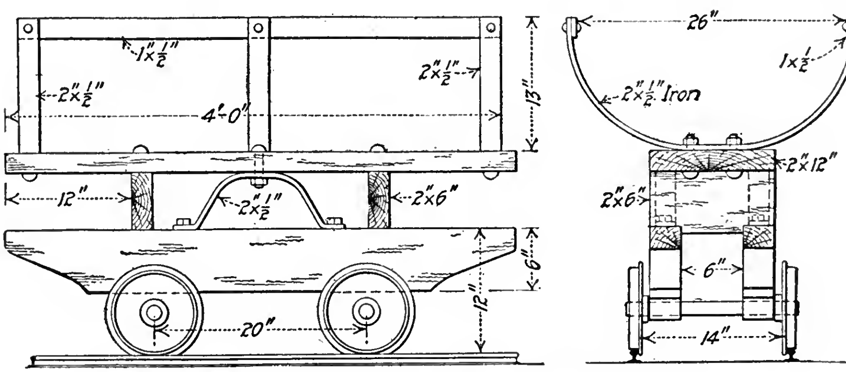


FIG. 354.—JOPLIN CAR FOR TRAMMING MINE SUPPLIES.

district, chiefly for transferring the ore from a subsidiary shaft to the mill hopper or bin, in some instances for hauling boulders up an incline to the dump, and in other instances for conveying dewatered tailings to the tailings pile. In all cases they are used on a flat grade, so that a front door is required to hold the load in place. The body of the car is made of 2-in. oak, lined on the inside with No. 12 plate and held together by iron

straps at the end and middle. The wheels are keyed to the axles and the axles rotate in bearing pieces of hickory. The bridle is fastened by side straps *D* to a crosspiece of round iron *E* at the rear end of the car. Guide pieces *A* are bolted to the sides near the front end, so as to guide the straps back down the sides of the car in case they move above the sides when dumping. Brackets *B* are also provided for the bridle straps to rest on while the car is being filled and while the cable is slack. As the car dumps by the raising of the rear wheels above the line of the track, the tread of the back wheels is made wider than that of the front wheels. This is effected by putting two separate wheels on the rear axle, the outside wheels on each side being loose. On this account, no special wide-faced wheels have to be cast for these skip cars. Because the grades are flat, the cars have to be fitted with a front door. This is latched by the

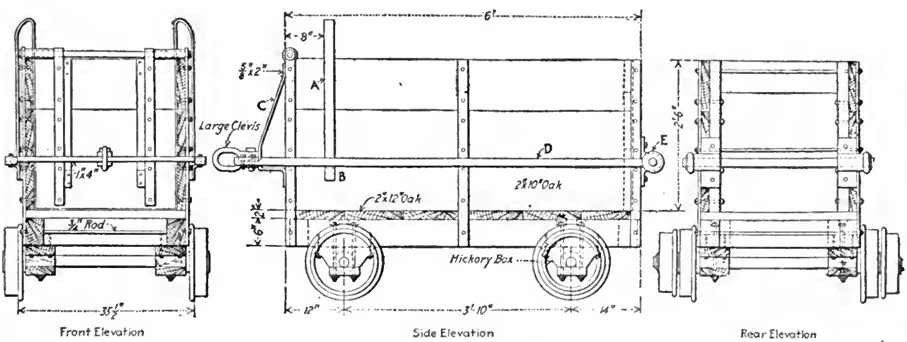


FIG. 355.—SKIP CAR USED ON THE SURFACE IN THE JOPLIN DISTRICT.

bridle which, after the car dumps, drops down on one or two bracket pieces *C*, bolted to the door of the car. The dump consists of two timbers put up at such an angle that the rear end of the car is raised about 4 ft. in a distance of about 10 ft., and owing to the slow speed with which the car is hoisted into the dump, no attempt is made to ease the shock by curving the dump track. The raising of the rear end of the car raises the cable and lifts the bridle off the brackets, unlatching the doors. When the car is lowered back out of the dump, the bridle drops as the rear end of the car falls and the door is again locked in place.

Roller Barrow.—The roller barrow shown in Fig. 356 has a capacity of 3 cu. ft., or about 1/4 ton of magnetite ore, and weighs, empty, 125 lb. This type of barrow is supplanting the ordinary wheelbarrow, which must frequently be used in mining portions of the flat-dipping orebodies at Mineville, N. Y. The wide roller, which will nicely ride a plank, together with the low center of gravity, obviates the usual strain in the wheeler's arms to keep the barrow from tipping sideways. The barrow is generally

dumped endways, the nose acting as the turning point, and braking is accomplished by lowering the handles until the roller boxes or rear end of the barrow touch the runway. The wooden roller, boxes and handles

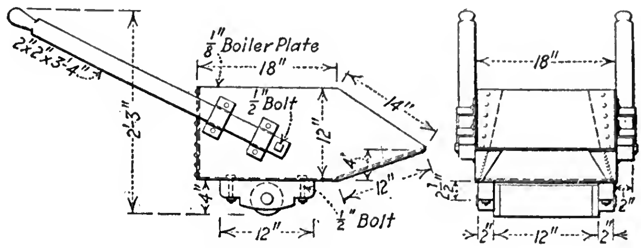


FIG. 356.—UNDERGROUND SINGLE-ROLLER BARROW.

are easily replaced when broken. The ⅛-in. boiler plate is cut from a single sheet, bent to slope and the laps riveted.

Truck for Lowering Timber.—The truck shown in Fig. 357 is designed for lowering timber into a mine through an inclined shaft; in this particu-

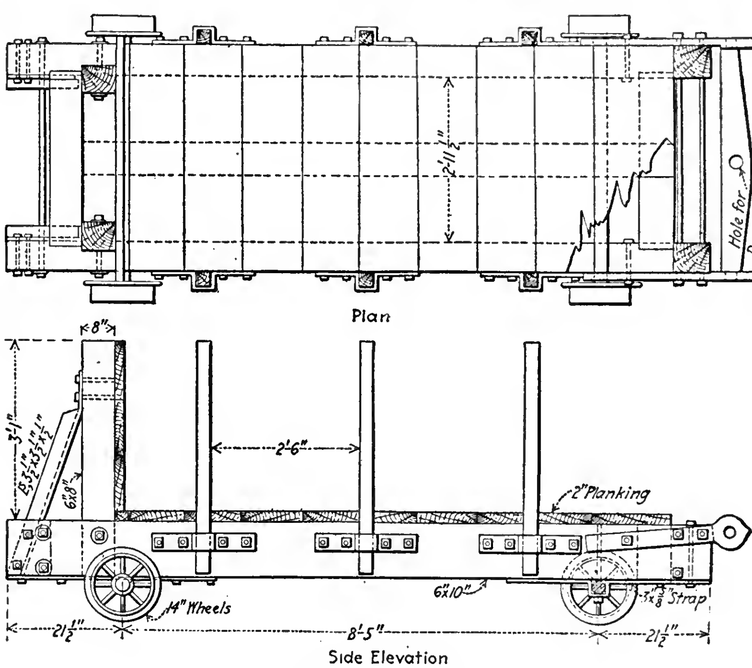


FIG. 357.—TIMBER TRUCK FOR 40° INCLINE SHAFT.

lar instance the shaft for which the truck was designed had an inclination of 40° from the horizontal. The truck can be readily made by the mine carpenter and blacksmith. Bail, bridle, wheels and axles, similar to those

of an ore skip, are used. The truck consists of a platform mounted upon wheels and axles. To the back part of the truck upright timbers are attached, against which the ends of the timbers being lowered rest. The timbers on the platform are held in place by the side uprights of which there are three on each side.

ACCESSORIES

Car Wheels for Sprag Braking.—In the development of a flat ore deposit with an extremely irregular floor, such as is found in the lead district of southeastern Missouri, heavy and changing grades are frequently unavoidable. Where animal haulage is used in such mines, some method of holding the cars in check is necessary. The ordinary brake beam does

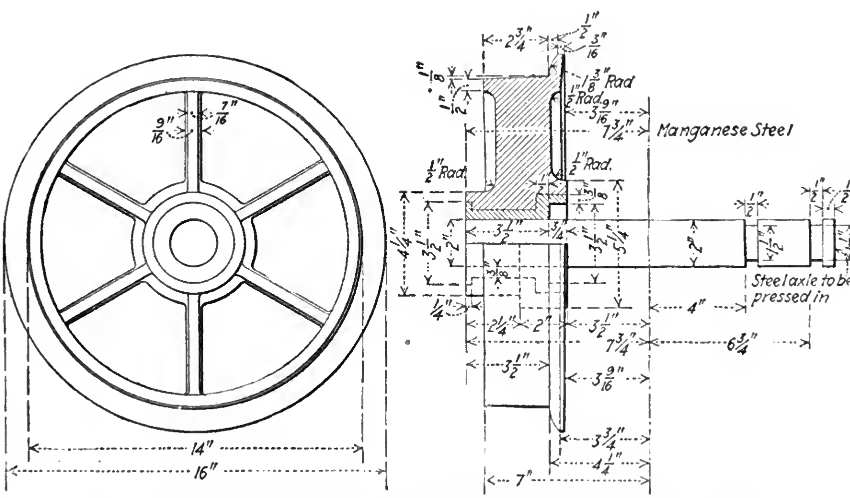


FIG. 358.—MANGANESE-STEEL CAR WHEEL USED BY FEDERAL LEAD CO.

not operate successfully on steep grades and the method sometimes used of making the animal help is hard on the animal. The method often followed in such cases is to design the car wheels with heavy spokes between which the driver shoves a pin so as to lock or sprag the wheel and cause the cars to skid down the grade with as many wheels locked as necessary. The driver picks up the sprag pins at the top of the grade and moves from car to car locking the wheels. On the return trip the pins are thrown off at the head of the grade. Cast-steel wheels were used for a time by the Federal Lead Co., but proved unable to withstand the severe shocks caused by spragging. This company has now adopted manganese-steel car wheels of the pattern shown in Fig. 358. These wheels are used on 1.4-ton cars and no trouble has been experienced by their failure.

Device for Retarding Speed of Cars (By John T. Fuller).—Fig. 359 shows the essential details of a simple device for checking or retarding the speed of cars used at the diamond mines in Kimberley, South Africa. It consists of a platform built of one or more pieces of timber, bolted or spiked to the ties between the rails, with a slope at each end as shown. Over this platform wrought-iron or steel plates 1/8 or 1/4 in. thick are spiked, bolted or screwed. The height of this platform above the rail is just sufficient to raise the wheels of the car free from the rail when the axles come into contact with the platform. The car travels the length of the platform on its axles, its speed being gently and gradually checked to any degree desired depending on the original speed and the length of the platform.

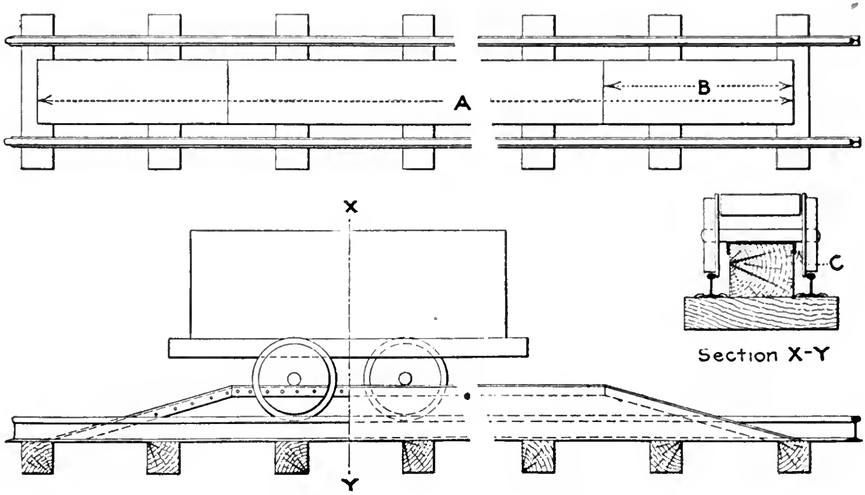


FIG. 359.—KIMBERLEY CAR CHECK.

The width of the platform will depend on the width *C* of the journal boxes. The length of the platform must be determined by experiment for each case; but this is quickly and easily done. The length *B* of the inclined portion of the platform is usually about 3 ft. and there have been used at different places platforms with the length *A* from 10 to 25 ft. When once properly adjusted this car check will be found not only simple and economical but absolutely automatic and indestructible under the roughest usage. This device can, of course, be used only where the car axles are free to revolve. Contrary to expectation, there has been experienced no trouble with bent axles by using this form of check.

Lever and Lock for Side-dump Car.—The typical car for underground electric haulage on the iron ranges has a gable bottom and side dump, holds from 2 to 3 tons and is mounted on a single truck. The devices for locking and releasing the side door are as numerous as the companies

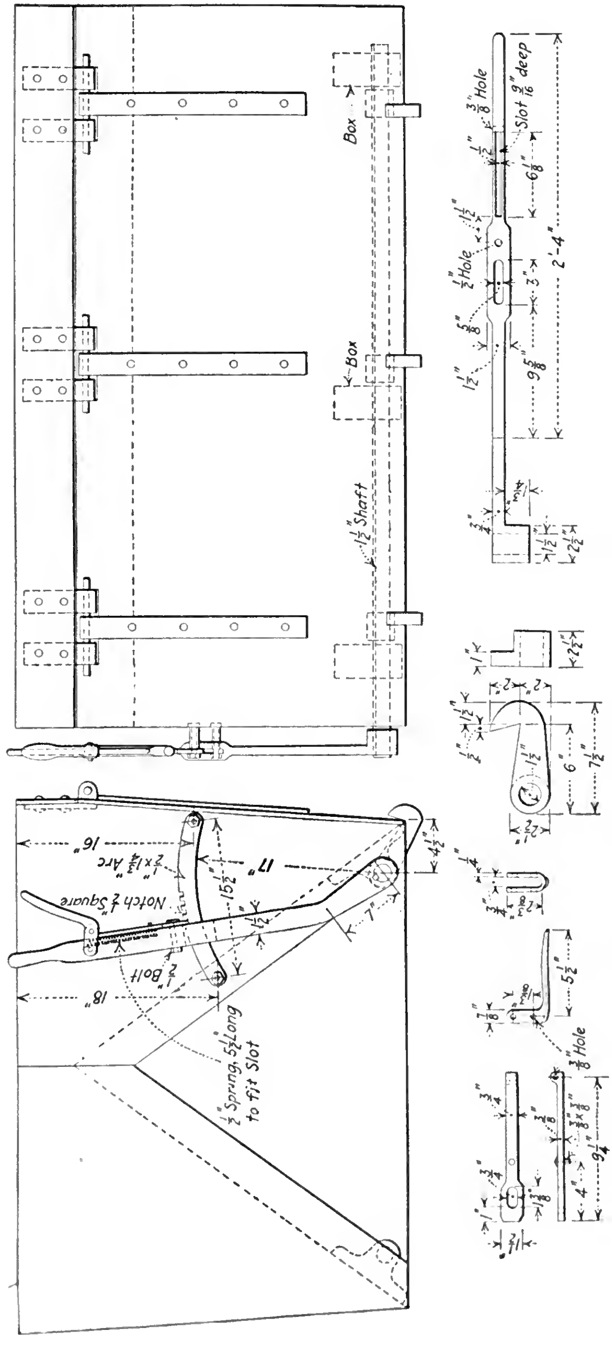


FIG. 360.—SPRING LEVER APPLIED TO SIDE DOOR OF CAR.

operating. One of the best is that used in the Oliver mines at Ely, Minn., illustrated in Fig. 360. Two levers with spring stops, similar to the brake and clutch levers on small hoists, are mounted on opposite ends of the car, so that one operates each door. As the train comes to the pocket, two men take position on opposite sides of the track and release the levers so as to open the doors as the cars pass, the operation being extremely rapid. The weight of the ore swings open the door so as to permit a free discharge and any material that sticks to the bottom is released by banging the doors vigorously. Each lever is keyed to a 1½-in. horizontal shaft extending the length of the car under the inclined bottom. This shaft works in three one-piece cast-iron boxes fastened to the underside of the

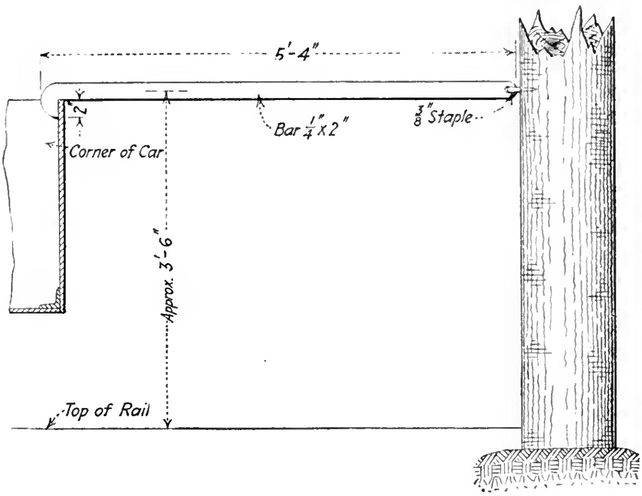


FIG. 361.—KENNEDY CAR GRIP.

car bottom in an inclined position. Three hooks are also keyed to the shaft and their ends engage the lower edge of the door. A handle, spring and sliding stop are fitted to the lever in the ordinary manner, the stop engaging notches in an arc which is held out from the end of the car by bolts and spacers and which passes through a slot in the lever. In locking the door, the lever is forced well along the arc and when the stop enters a notch, there is enough spring in the lever, keys and shaft to keep the hooks tight against the door. The arrangement and dimensions are shown in the drawing.

Latch for Holding Car During Loading.—At the Kennedy mine on the North Cuyuna range, the device represented in Fig. 361 is in use for holding cars while loading from a chute, in cases where the grade might otherwise cause the car to run away. The drawing is an elevation taken diagonally across the drift. The bar is held to a drift post by a staple

and is of such a length that when it is caught in a corner of the car, the latter is properly positioned for loading from the chute.

Car-bottom Straightener.—In loading cars from stope chutes, especially where stoping is done by the shrinkage system, the falling of large pieces of ore batters the bottoms of the car boxes out of shape. These car bottoms must be straightened from time to time, and in Fig. 362 is shown a device which works much better than the time-honored sledging method. A frame, 3 × 7 ft. in the clear, is constructed of a 6 × 6-in.

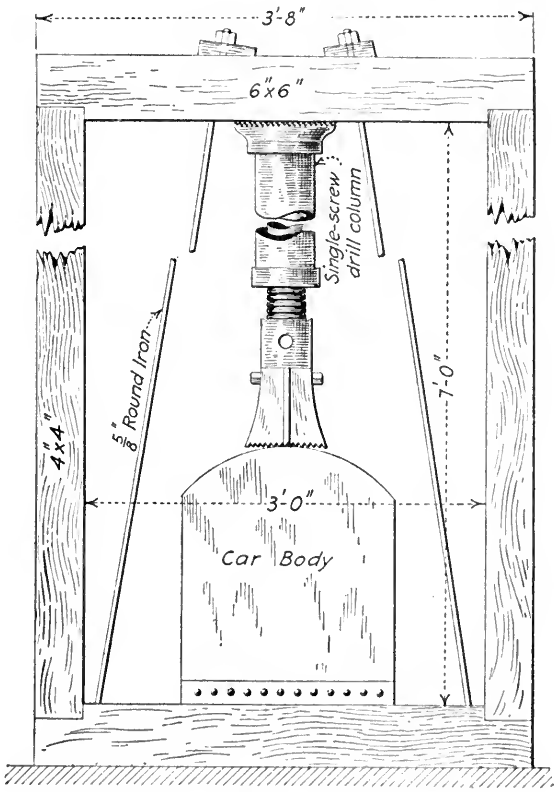


FIG. 362.—SINGLE-SCREW COLUMN FOR STRAIGHTENING CAR BOTTOM.

sill, two 4 × 4-in. posts and a 6 × 6-in. cap, the whole held securely together by two ⅝-in. tie-rods, and firmly braced. The car box, the bottom of which needs straightening, is removed from its truck and placed under the frame, bottom side up. A single-screw drill column is then set up as shown, and pressure brought to bear on the box. By applying this pressure in two or three places, if necessary, the bottom is pressed back into shape without the denting and local strains which sledging would cause.

XI

TRACK

Track Arrangement—Laying Track—Switches, Etc.

TRACK ARRANGEMENT

Track Work in a Minnesota Mine (By E. W. R. Butcher).—Fig. 363 shows some details of underground track work and one of the motor turns used in the mines of the Republic Iron & Steel Co., on the Mesabi range. A right and left turn is shown in 1 and 2. As a rule, the location of a turn is determined before the drift is driven and the necessary sets are put in place to make the turn when required. Props are placed under the ends of the two caps resting on the opening set until that turn is to be driven. A 9-ft. by 6-in. post is used under both of these caps and on either side the posts of each set are shortened 6 in. until an 8-ft. post is reached, which is the length of post used in motor drifts. When the opening set is placed in position, a point is placed on the set and on the 10-ft. by 9-in. set and with this line the rest of the turn is put in with the aid of the other dimensions shown. In 3, 4 and 5 are shown the track layout and frog details used in connection with a 25-ft.-radius timber turn. The frog is designed so that it can be used for either a right or a left turn. The stub switch has given better satisfaction for underground work than the point switch. The latter caused considerable trouble by dirt getting between the wing rail and track, which interfered with its closing. In 6 to 10, inclusive, are shown the details of switch stand and tie-rod connections.

Tracks for Loading Station.—In the mines of the St. Louis Smelting & Refining Co., in southeastern Missouri, a three-track approach to the shaft is used, since the ground stands well and the shaft station may be made extra wide. The 1-ton cars are drawn by an electric motor in trains on tracks of 24-in. gage. The loaded cars are pushed in trains ahead of the locomotive, on to the middle track, while empty cars are pulled from the cages and are generally put on the outside tracks. The drift end of the three-track approach, which is 100 ft. long in some instances, has to be fitted with a three-way switch, as shown in Fig. 364, at the point where the two tracks make off from the single track. The curves on this track can be made easy. The arrangement of the track and the frogs is shown. At the loading end of the three-track approach,

the crossovers, as shown in Fig. 365, have to be made abrupt, so that the switching from one track to the other may be done in the minimum of

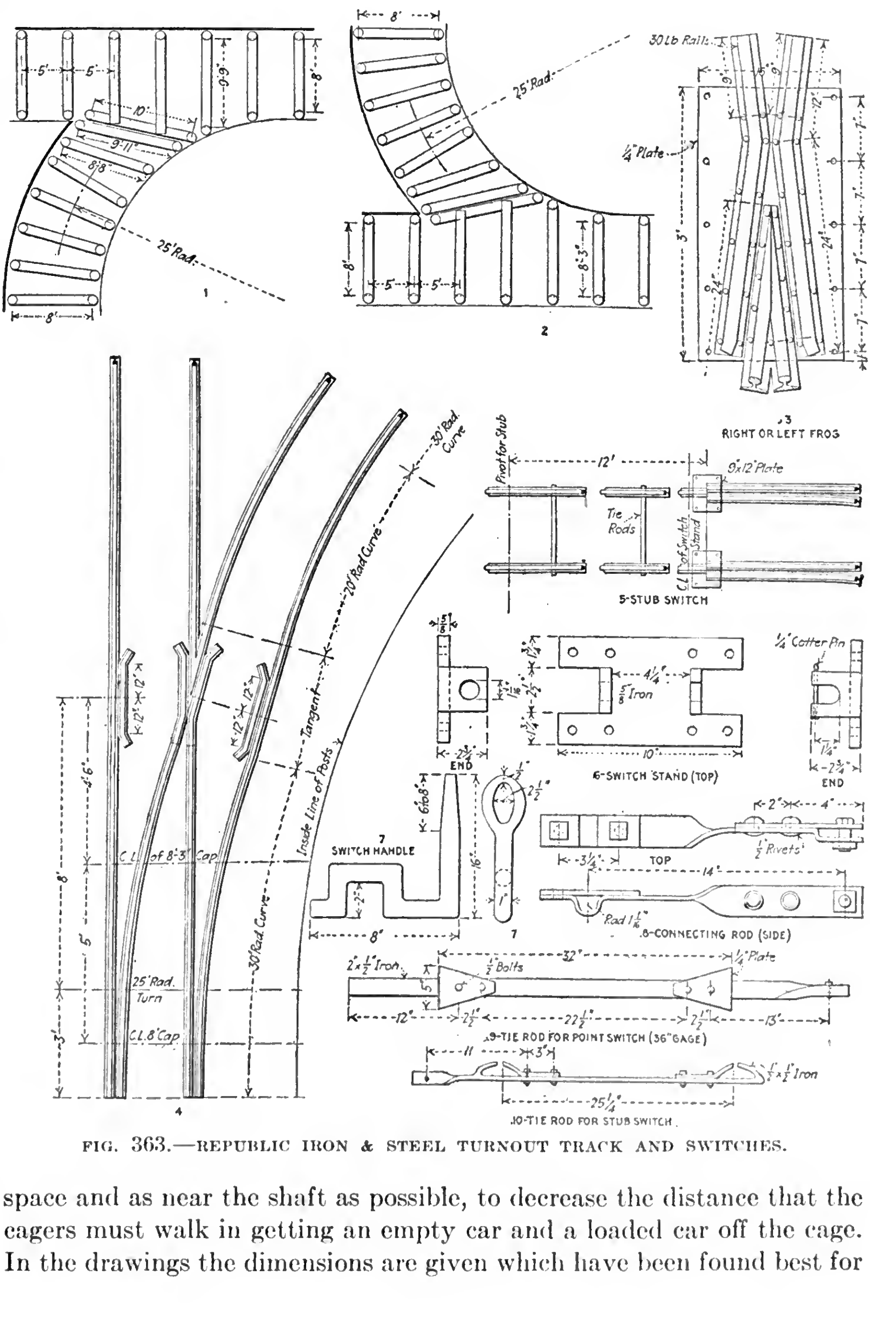


FIG. 363.—REPUBLIC IRON & STEEL TURNOUT TRACK AND SWITCHES.

space and as near the shaft as possible, to decrease the distance that the cagers must walk in getting an empty car and a loaded car off the cage. In the drawings the dimensions are given which have been found best for

loading cages at greatest speed. At first a distance of 13 ft. from the shaft to point of middle frog was tried, but this was found to be too great and was shortened to 11 ft. This has been found to be just about right, the curve beginning almost as soon as the cars clear the shaft. The distance between outside rails in the three-track straightway is 11 ft. This gives plenty of clearance between cars. In the loading crossovers a kick

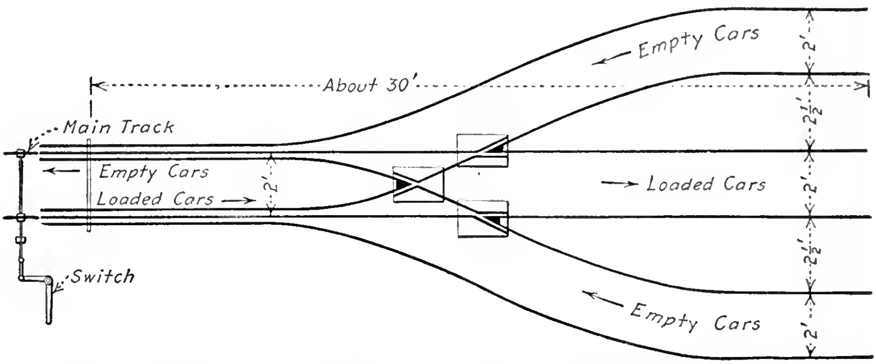


FIG. 364.—DRIFT END AND THREE-WAY SWITCH FOR TRIPLE-TRACK LOADING STATION.

switch is needed to join the middle track with the track that connects it with the shaft, shown in Fig. 365. The cagers throw this with one foot and after running the empty car out on the outside track on the same side of the shaft that it came from, cross over to the center track to get the loaded car.

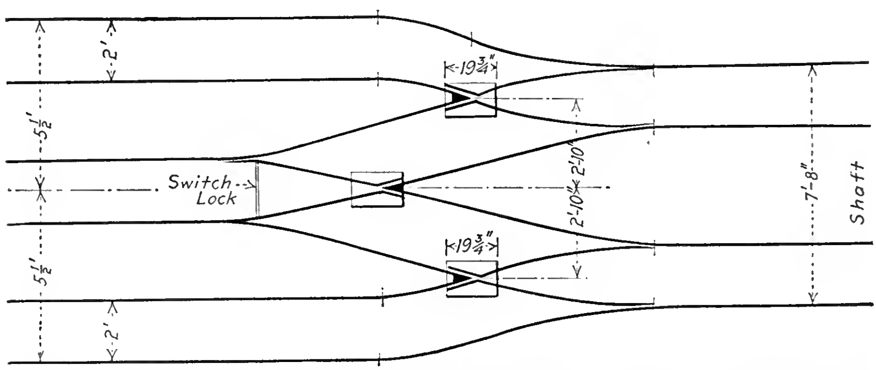


FIG. 365.—CONNECTIONS BETWEEN TWO-COMPARTMENT SHAFT AND THREE-TRACK STATION.

Economical Incline Track Arrangement.—The working shaft of the Sterling Iron & Ry. Co., at Lakeville, N. Y., is in the orebody and follows approximately the flat-dipping foot-wall. The foot-wall is extremely irregular and while some of the rolls are followed by the shaft, it was

necessary to cut through some of the sharper ones. In order to reduce the width of these cuts and also to save a certain amount on rails and ties, a rather ingenious method of arranging the tracks was resorted to, as shown in Fig. 366. Hoisting is done in balance from several levels so that provision had to be made for the skips to pass. From the top of the headframe to the 800-ft. point, three rails are used. This is not an unusual method on inclined planes and results in saving the cost of one rail, a

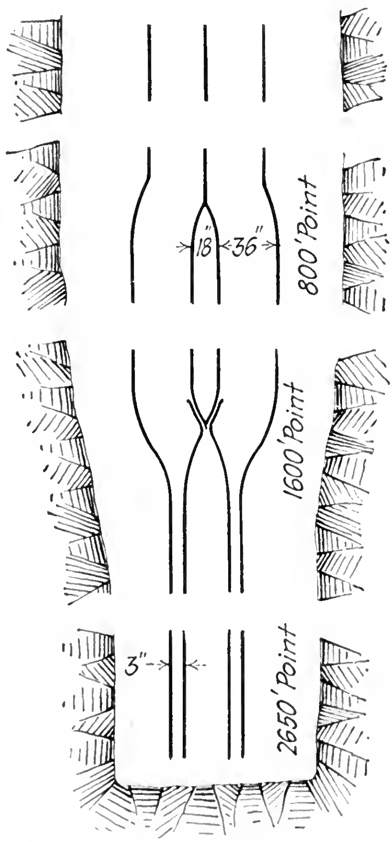


FIG. 366.—ECONOMIZING EXCAVATION IN INCLINE DOUBLE-TRACK SHAFT.

certain amount in the length of the ties and some rock cutting. At the 800-ft. point a change is made to four rails, two complete tracks, in the manner shown, no switch being necessary. The tracks are continued to the 1600-ft. point and the two skips can pass anywhere between these points. From the 1600-ft. point down, it was desired to carry the shaft as narrow as possible, but the management was unwilling to risk possible accident from the use of the switch which would have become necessary

if a single track, that is, two rails were there used. The safety of a hand-operated switch at such a point depends on the memory of the operator as to which track the last skip was sent over, and an automatic switch is liable always to get out of order. The scheme illustrated was conceived and has given excellent satisfaction. While it uses four rails, no increase in the length of the ties is necessary and not much in the width of the cut over that required for a single track and the saving over the three-rail system used in the upper part of the shaft is appreciable. No switch is required at the 1600-ft. point to change to the two-track portion. The arrangement continues to the bottom of the shaft at 2650 ft. below the collar. There are no compartments in the shaft, no timber whatever being required with the excellent hanging; the shaft is quite open, really a part of the stopes in places.

Track Spreader and Guard-rail Bracket.—The type of track spreader shown in Fig. 367 is particularly useful for keeping the rails to gage on timber stringers which have become partially rotten, so that they will not hold a spike securely. Of course, the surest remedy is to replace the stringers, but often the use of a track spreader will keep the skipways in

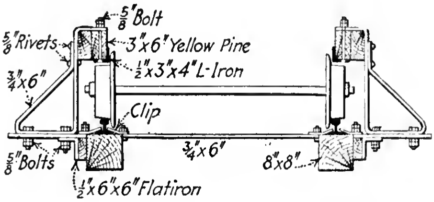


FIG. 367.—TEMPORARY REPAIR FOR ROTTEN SKIP STRINGER.

commission until holidays, when stringers can be replaced without interrupting the mine output. A $\frac{3}{4} \times$ 6-in. flat iron, bored, for $\frac{1}{2} \times 2\frac{1}{2} \times$ 4-in. clips, is driven under both rails after sufficient rotten timber has been cut away, as seen in the illustration. The clips are then bolted to the flat iron. This will keep the track to gage, even though the rails may move to the sides of the stringers. To keep the rails centered on the timber, $\frac{1}{2} \times 6 \times$ 6-in. clips may be bolted to the bottom of the spreader and wood blocks can be driven between the clip and the stringer, the stringers being kept the proper distance apart by the usual $3 \times$ 6-in. spreader held by a girt rod. To save the cable, rollers are used near the iron spreaders. Brackets of $\frac{3}{4} \times$ 6-in. flat iron, to hold the back or guard rail, may be bolted to the projecting ends of the track spreader. The same type of bracket on new skip-ways is made longer by an amount equal to the depth of the stringer, and bolted to the wall plates, which are usually introduced at 15-ft. intervals.

Track Curves in Top-slice Rooms.—In the top-slice method of ore extraction as practised on the Mesabi, successive rooms are opened off the crosscut, extending 10 ft. one way, and 40 ft. the other. The track from the crosscut is turned into the long side of the room only. For the first room mined from any crosscut, a curve is fitted into the track such as represented by the portions of the rails *A*-*A′*, Fig. 368. When the room is mined and caved, the track is taken out and it is desirable to use the same curved portion for laying track into the next room. It is only rarely that the break in the crosscut rails is such that the curve will fit. The difficulty is overcome by Captain James Rosewall of the Harold mine by using two switch points, as shown. The curve rails are laid, the portions *B* of the crosscut rails are spread apart and the switch points fitted in to

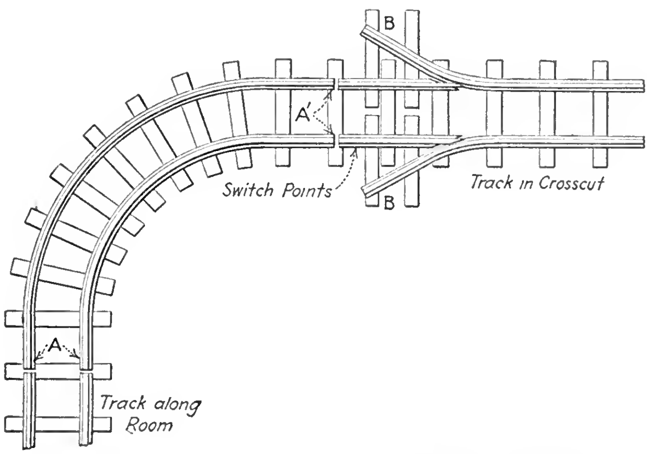


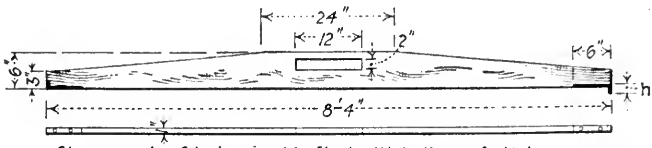
FIG. 368.—FITTING TRACK INTO ROOM FROM CROSSCUT.

make a tight and smooth joint. The switch points used, of course, are such as would be used for a split or fixed switch and not for a stub switch. The trick is one taken from openpit practice, where in changing tracks it is often difficult to get joints to match without recourse to this device.

LAYING TRACK

Convenient Grade Stick (By Edward H. Orser).—The grade stick shown in Fig. 369 is a handy form of one of the most useful tools in the trackman's equipment. It consists of a 1 × 6-in. board fitted with two iron angle shoes at the ends, the middle part of the top of the board being straight and true and parallel with the bottom. A hand-hole is cut as shown and the ends of the top are beveled off to reduce weight. The shoes on both ends are the same, but are placed in opposite positions the

short leg turning up in one case and down in the other. This short end is made of a length to give the exact grade desired; for example, if a 0.5 per cent. grade is wanted and the grade stick is 8 ft. 4 in. long, the length of the short leg of the shoe will be ½ in. In drifting, the left end is kept ahead. When a new section of track is to be laid, the back end of the grade stick is set on one rail and a hand level placed on the flat top. The new rail is raised until the bubble is centered. If on straight track, the other rail is set by leveling across. If on a curve, the proper allowance for raise is made in the outside rail. If the level is out of adjustment, it



Shoes made of ¼" iron, set in flush with bottom of stick.
h, varied to give grade desired. Holes spaced 2" apart, ¼" diam.

FIG. 369.—IRON-SHOD GRADE STICK FOR MINE TRACKS.

should be reversed and the new rail lowered or raised so as to split the difference.

Angle-iron Track Gage.—A serviceable track gage can be made from a piece of angle iron as shown in the accompanying drawing, Fig. 370. One leg of the angle is cut away flush with the face of the other leg and back from the ends 3 in. The remaining length of this leg gives the proper gage for the track and the 3-in. projections of the other leg rest on top of the rails. A ½-in. round handle riveted on the upper or longer leg of the gage makes it complete. With a little use track gages constructed

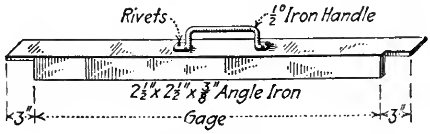


FIG. 370.—STIFF GAGE-IRON FOR TRACK LAYING.

of wood soon become so badly battered that they are practically useless and those made of iron are easily bent, thus giving in both cases a distance shorter than the gage desired for the track being laid.

Bending Rails with Screw-jack (By A. Livingstone Oke).—There is illustrated in Fig. 371 a method for bending rails, consisting of the use of a screw-jack in conjunction with three upright posts planted in the ground to form the apexes of an isosceles triangle, as shown. The rail to be bent is placed across two of the posts and the jack is footed against the third. To facilitate the manipulation of the jack and rail, it is desirable to use

three blocks of wood, the longer one slightly inclined on its upper surface to form a rest for the jack, and the other two notched to receive the side of the rail and give a steady support. To prevent the movement outward of the tops of the posts, three links are made with eyes at the extremities, which drop over round iron dowels driven firmly in the tops of the posts. The lower ends of the posts are sunk from 1 to 2 ft. in the ground. This device has been found useful in mines, as it is possible to place posts arranged in this manner at entrances and at suitable points underground,

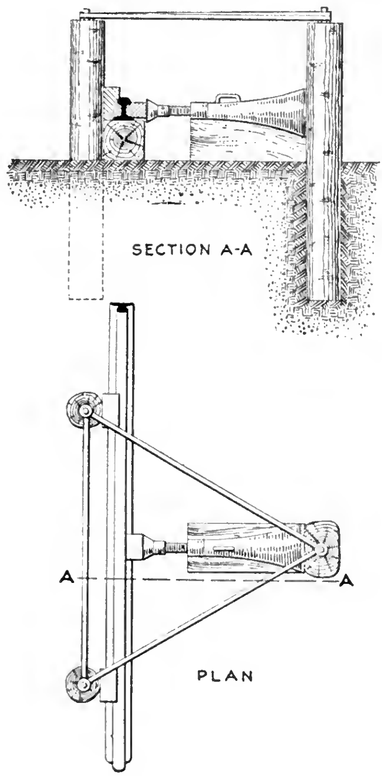


FIG. 371.—POSTS AND RESTS FOR SCREW-JACK RAIL BENDER.

where frequent rail-bending is necessary, and thus avoid the transportation of the usually heavy jim-crow, the lighter screw-jack being all that is required to carry about.

Notched-log Rail Bender (By Charles F. Spaulding).—A sturdy and handy device for bending rails can be made of two logs, a 6 × 6-in. piece and a screw-jack, as shown in Fig. 372. The logs are laid side by side and two notches cut at opposite points in each. In one pair of these, the 6 × 6-in. piece is wedged. In the other, the rail is laid. The dis-

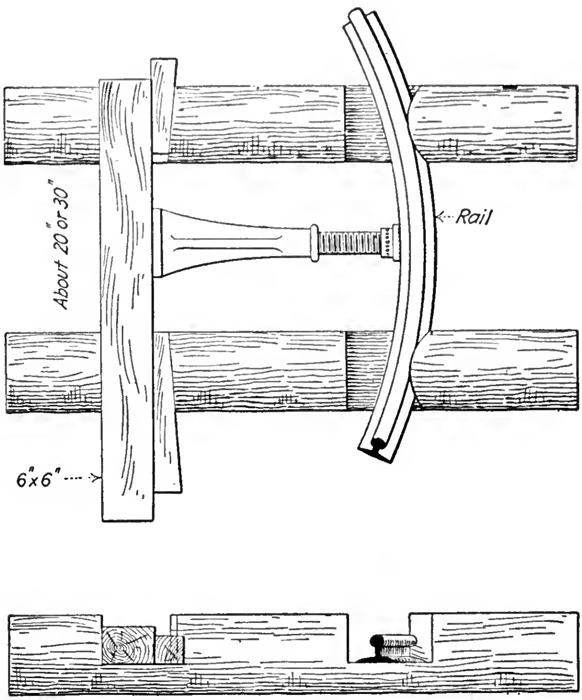


FIG. 372.—RAIL-BENDER OF SCREW-JACK AND ROUND TIMBER.

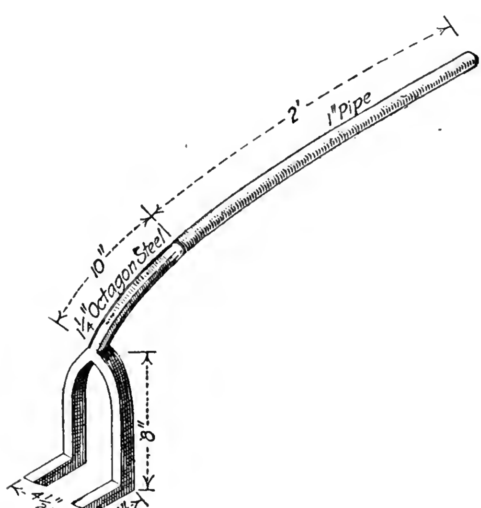


FIG. 373.—LEVER FORK FOR HOLDING TIE TO RAIL.

tance between notches along the logs should be such that a 1⅛-in. or 1¼-in. jack, footed against the crosspiece, will easily reach the rail. For curves of different radius, the logs can be laid a greater or less distance apart.

Rail and Tie Holder.—In laying or repairing mine tracks, it is not always possible to have the rails resting upon all the ties before spiking, due to the unevenness of drift and crosscut floors. To overcome the difficulty of spiking under such conditions, the tool shown in Fig. 373 was devised. One end of a 2-ft. length of 1¼-in. octagon drill steel was split for 1 ft. and shaped into a fork, the end of each prong for a length of 4 in. being bent forward at a right angle; the other end of the drill steel was drawn out so that it would just slip into a piece of 1-in. pipe; this pipe serves as a handle and lever. By placing the fork over a rail and slipping the prongs under a tie, the latter may be held up firmly against the rail while being spiked.

SWITCHES, ETC.

Switches and Crossings (By D. W. Jessup).—In underground track work, where there is heavy traffic, the switches in general use are the stub and split switches and their modifications. These switches are operated in various ways; usually by means of a stand and lever, a crank and lever, a toggle joint, or a target. Where light traffic is concerned, especially a one-car traffic, the switch is operated by the hand or a kick of the foot and gives satisfactory results. In laying track it is important to remember that the frog should be elevated about ¼ in. above the rails, as this throws the car against the rail opposite the frog and prevents the car wheels from catching the frog.

Fig. 374, at 1, illustrates a typical stub switch with a turnout *BB* from the main track *AA*. The switch points at *CC* are held together by means of a bridle *D* and fit into slots as shown in 2; with a broader gage more than one bridle is often used. The bridle is moved to and from *AA* to *BB* by various lever methods, principally by those given above. The throw or movement of the switch rails for an 18-in. gage is about 1½ in. and a ⅛-in. space is left between the switch point ends to allow for easy shifting. Allowance for movement of the switch rails *CC* must be made; these rails are not spiked for a distance of 12 or 15 ft. back from the switch points, and to replace the spikes, clamps *E* are placed about every 3 ft. from the points. Underneath the switch points is placed a long, solid 6 × 6-in. tie, which extends to the switch levers, and facilitates the movement of the rails. To prevent wearing, a strip of sheet iron 3 or 4 in. wide is fastened to the tie underneath the points.

The typical railroad split switch is but little used underground in metal mines, except with a track of broad gage, and in mines with a large

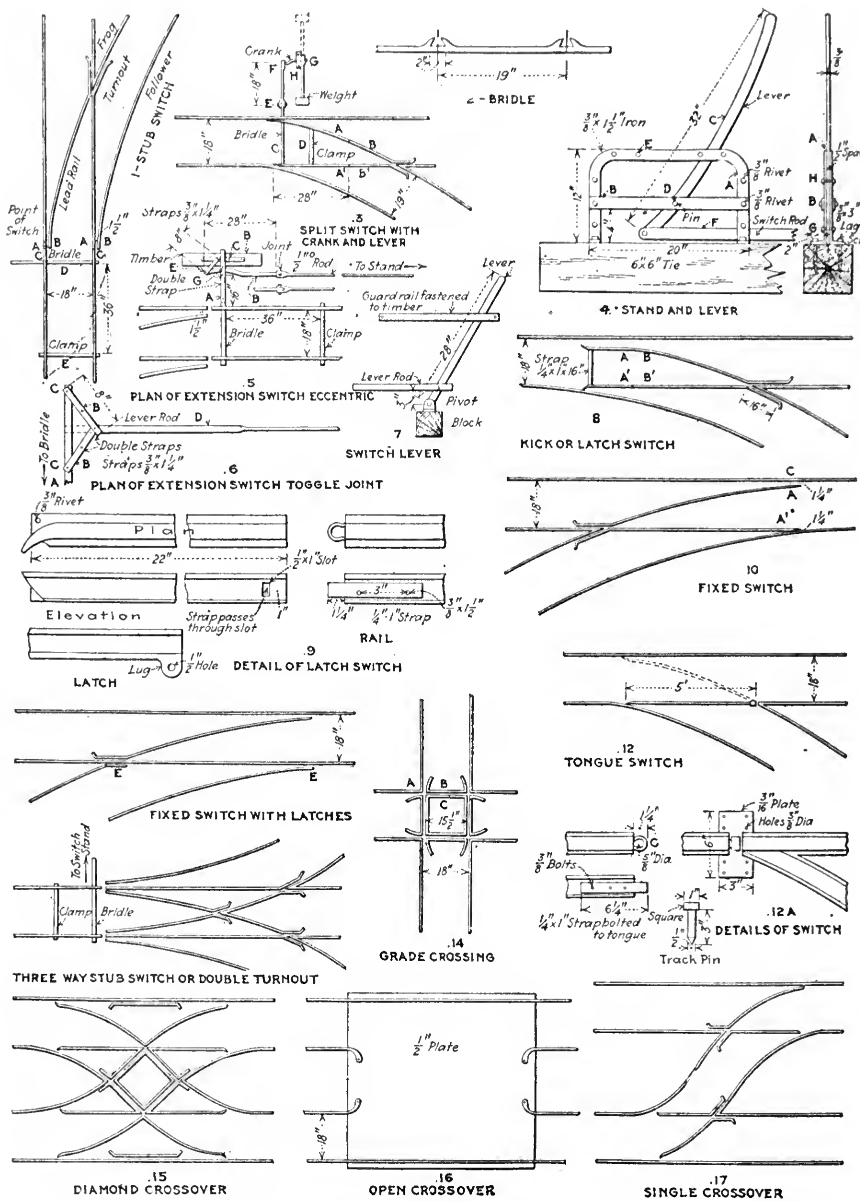


FIG. 374.—SWITCHES AND TRACK EQUIPMENT ADAPTED FOR MINE USE.

traffic, since it is expensive to make and lay, and is not adapted to short turns. A modified split switch is used as shown in 3. Two latches AA' are fastened to the rails BB', and are held together by a clamp D and a bridle C. The latches vary in length from 24 to 36 in. and are drawn to a point at one end, flattening the outside and leaving the inside about normal, or slightly curved inward, the length of the taper depending upon the length of the turnout. The bridle is operated by one of the different switch levers.

The stand-and-lever switch is found in operation in many mines and affords satisfaction. It has the advantage over the crank and lever in that it offers some latitude in the throw of the switch, as is often demanded in the double turnout or in cases where the rails spread and the switch points are not in line, while the crank and lever offers but one distance of throw. The stand is spiked to a 6 × 6-in. tie which extends under the switch points, as shown in 4, and is placed about 18 in. from the track, allowing sufficient space so that a car running on the track will not strike the lever arm or stand. The latter is about 20 in. wide and 12 in. high, consisting of a double frame A made from $\frac{3}{8} \times 1\frac{1}{2}$-in. iron, the halves spaced $\frac{1}{2}$ in. apart, and riveted together at H and G. About 2 in. of each leg is turned out and fastened to the tie by lag screws as shown, and a double strap B is riveted across the frame about 4 in. above the bottom. The lever arm C passes between these straps and is fastened to them by a loose bolt or a pin at D, about which the lever is pivoted. The length of the lever is from 24 to 36 in. and the lever is bent inward so that a passing car will not strike it; the lever is held in place by a pin passing through the holes E in the stand. The lower end of the lever is fastened to the switch rod by a bolt or pin, the rod extending to the bridle. The stand is sometimes made so that the legs are spiked to the sides of the tie instead of the top, but it then lacks stability.

The crank-and-lever-arm method of movement is shown in 3. An arm or switch lever about 18 in. long is fastened to the bridle at E, and the other end is fastened to the crank shaft at F. The throw of the crank should be exactly equal to one-half the throw of the switch. The length of the crank shaft is about 4 in.; it is attached to a block or tie by means of a strap H. The lever arm is thrown by hand. Its length should be such that when thrown toward the track it will not touch it. The weight on the arm is 5 or 10 lb. A piston rod and chuck from an old machine drill will answer the purpose.

At times it may be convenient to have the lever situated at some distance back from the switch points, and of the several devices an eccentric or a toggle joint is often used. The eccentric is shown in 5. A switch rod consisting of a double strap A about 24 in. long and extending out from the bridle to a piece of timber B, passes over a lateral strap C, with-

out being fastened to it. The strap *A* is riveted in two places *G*, spaced 3 in. apart and about 16 in. from the rail, the eccentric of the lever rod *D* passing through the space. A short arm *E* about 8 in. long is attached by a pin or bolt to the lever rod *D*, and also to the lateral strap *C*. The eccentric of the lever rod is equal to the throw of the switch, 1½ in. The rod is jointed and extends to the switch stand. By operating the lever, the eccentric moving between the rivets *G* causes the switch lever to move the track from either point of switch.

The toggle joint, 6, is of simpler construction than the eccentric switch, and is perhaps more satisfactory. A switch lever *A* connects the bridle to one of the toggles *B*; the toggles consist of a double strap about 8 in. long fastened at *C* and to the lever rod *D* by pins or bolts. By operating the lever rod *D* the toggles are pushed in or out, causing the bridle to move at the switch points. The lever may be of simple construction as shown in 7.

The following switches are designed mostly for one-car traffic; they give satisfaction, and are easy of manipulation, being adjusted by the hand or foot. The kick or latch switch is recommended as being one of the most efficient of the lighter switches, due to its short length, its durability, and its easy adjustment by a kick of the foot or a movement of the car, the carman not finding it necessary to advance ahead of his car to adjust the switch unless running toward it. The only disadvantage is that if repairs are needed it may be necessary to remove the whole switch to the blacksmith shop. The switch, shown in 8, consists of two short switch points or latches *A*, 20 to 24 in. long, fastened to the rails at *B* by means of a looped strap; the points are tapered and turned slightly outward. A bridle strap 14 to 16 in. long is riveted underneath to the points of the latches, the holes through which the rivets pass being of larger diameter than the rivets, to permit the movement of the switch. The outside latch is 1 or 2 in. shorter than the inside latch, its length depending on the angle of the turnout. The looped strap fastening the latch to the rail is made of $\frac{3}{8} \times 1$-in. iron, as shown in 9, and is bolted to both sides of the rail, the loop first passing through the $\frac{1}{2} \times 1$-in. slot cut in the latch. The latches may also be fastened as shown in 12, the strap being bolted to the latch, through which a pin is driven into the tie. This method is unsatisfactory as the pin will pull out and cause derailments. Another and better method of fastening the latch is shown in 9, a lug being made on one side of the latch through which a hole is bored and a pin of smaller diameter driven through the lug and into the tie.

The switch shown in 10 is not movable and has no movable parts that will wear, as all of the rails are spiked to the ties. Sufficient space, about 1½ in., is allowed between the main track and the rails to admit the passage of the car wheels. The direction of the car is controlled by the car-

man who throws the car to either track by a twist; this is easily done with an empty car but with a loaded car it is troublesome. This switch is used to advantage when the loaded cars are running away from the switch. If the car wheel has a groove or double flange worn on its face, the car may tend to follow in the direction of the turnout. The railpoint A' should be in line with the main track, and as the point A may cause derailments, the rail at C is made lower than at A'; then the weight will cause the car to crowd closer to C and not tend to derail. The point at A should be slightly higher than the rail at C, which will assist the carman in throwing the car in the direction of the turnout. The length of the lead rail is usually from 5 to 7 ft., depending on the angle of the turnout.

A special form of fixed switch is sometimes used, giving an unbroken main line, as shown in 11, requiring the use of two hinged latches E, which are about 4 in. long and are fastened to the turnout rails. When it is desired to run the cars over the turnout, the latches are placed over the main line, a flange on the under side preventing their slipping; and when running on the main line the latches are swung out. This form of switch is not commonly used; it is troublesome, as the latches may slip, always takes time to adjust, and may accidentally be left on the main line, causing derailments.

The tongue switch, shown in 12, is used extensively with tracks over which there is light traffic, but it is not recommended for heavy traffic, though it is often used for such. Derailments frequently happen due to an open switch, or the pin pulls out that holds the tongue, or the tongue may turn over on its side causing cars to drop in between the rails. The advantage of this switch lies in its simple construction, the little repairing required, and the ease of laying. The movement of the car will not throw the switch as it will the kick switch, and it is necessary for the carman to advance ahead of the car to move the tongue. The details of the switch are shown in 12A. The point of the tongue is tapered and curved slightly outward; a strip of sheet iron is nailed across the tie over which the tongue point moves.

The three-way switch or double-turnout switch is used where two crosscuts are driven on opposite sides from a main drift, and it is desired to run tracks in the crosscuts from the drift. In this case the stub switch will be used to best advantage. It is operated by a stand and lever which allows the latitude of movement of the switch rails demanded by the nature of the switch, as illustrated in 13. To allow for the shift of the switch rails, they remain unspiked for a distance of 15 ft. or more.

Crossings are used where tunnels or drifts intersect at right angles and it is desired to continue the tracks in the one direction without turning on the other tracks. The crossing shown in 14 consists of four ordinary 90° crossing frogs A and the four inner crossing rails B which are a continua-

tion of the tracks; the latter have a length of 15½ in. leaving a space of 1¼ in. between the point of frog and the point of crossing rail. An inner guard rail or square section *C* is often used, affording a more satisfactory track. If the traffic is much heavier over one of the tracks, and it is desired to have the rails remain unbroken, then the track over which there is lighter traffic is raised 1½ or 2 in. above the other track, and hinged latches 4 or 5 in. long are swung over the main track to connect with the inner crossing rails when the lighter traffic track is to be used. When not using this track the latches are swung back again leaving the main track clear. But there is always the disadvantage of misplaced latches which give trouble.

A double crossover from one track to a parallel track is sometimes demanded. In 15 is illustrated the diamond crossover, one of the many kinds in common use. The switch points are fixed, there being no movable parts, and the direction of the car is controlled by the carman throwing his car. If desired, latch or tongue points may be used, operated

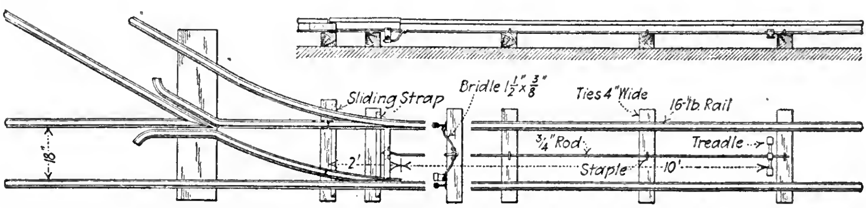


FIG. 375.—SEMIAUTOMATIC SWITCHES FOR MINE TRACK.

by hand or by switches, which will probably be necessary if the cars are ran in trains. The crossover is somewhat intricate for the average trackman to put down correctly, and at times is also troublesome to use. It may be done away with entirely with a one-car traffic, and a large piece of sheet iron substituted as shown by 16. The inner rails are cut out leaving only the two outside rails, and if the carman wishes to cross over he throws his car by a twist and passes to the other track. If he wishes to continue on the one track he twists his car toward the outer rail. The inside rails are flattened at the end, curved outward, and screwed down to the iron sheet. This open crossover may be substituted for the three-way switch, doing away with the guard rails and frogs. A single crossover is shown in 17; it has but few parts, is simple, and gives no trouble. The switch points can either be fixed or used as a kick switch.

Treadle-operated Switch (By A. H. Bromly).—Fig. 375 shows a switch which can be operated by the trammer as he passes with his car and before the car reaches the switch itself. As shown, it is applied to a track with 18-in. gage, the switch points being 24 in. long. The fish plates fastening the points to the fixed rails have only one bolt through the points and are

left loose on the point end to permit the necessary play. The bridle connecting the two points is bolted to the flanges and bent downward as seen in the cross-section. A small strap is riveted to the bottom of this so as to form a bearing for the end of a $\frac{3}{4}$-in. round rod. This rod lies between the rails for about 10 ft. and is held to the ties by staples so as to make a series of bearings. The end which operates in the bridle of the switch

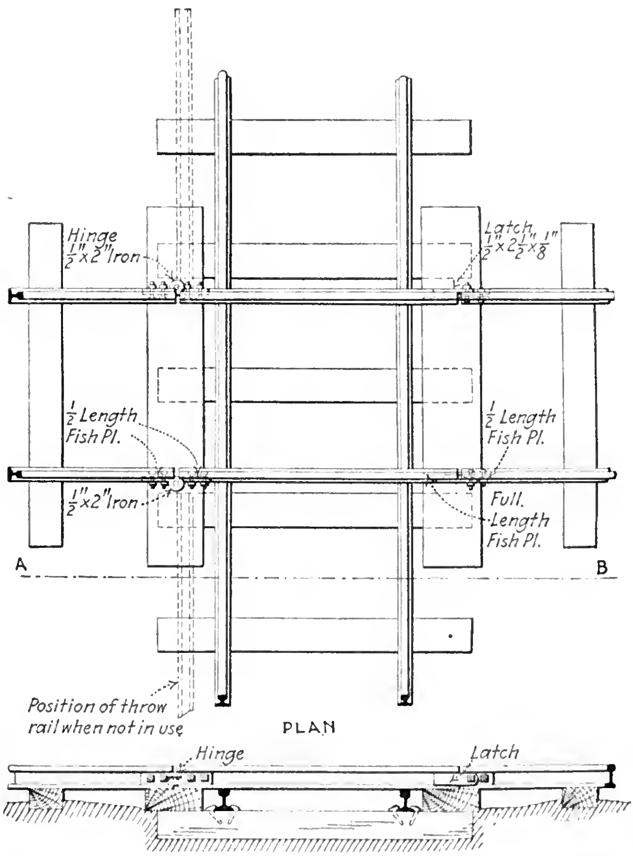


FIG. 376.—SWINGING RAIL CROSSOVER.

points is bent down so as to form a crank end. The other end is rigidly connected to a two-winged treadle lying between the rails, as shown. The trammer steps on the proper wing of this treadle as he approaches the switch and the latter is thrown as desired. The car will take the same side of the switch as the side of the treadle on which the trammer steps. The points should be provided with iron straps to slide upon.

Temporary Crossover.—In surface or underground work, conditions may exist where two tracks cross, without there being any necessity for switching from one to the other. If traffic is light or only temporary on one track, it is often advisable to provide it with a movable crossover, leaving the other track unbroken. In such case it is brought through at a slightly higher level so that its rails rest on the tops of the permanent rails. These rails at the crossing are hinged at one end and latched at the other and can be swung out when not in use, thus leaving the lower track unblocked. The construction of the hinges is shown in Fig. 376. They are held out by half lengths of fish-plates used as fillers, and are themselves of ½ × 2-in. iron. The latches are also held out from the fixed rails by fillers and are of sufficient width to bear on the head of the rail. They are of ½ × 2½ × 8-in. iron. The inside of each fixed rail is provided with a

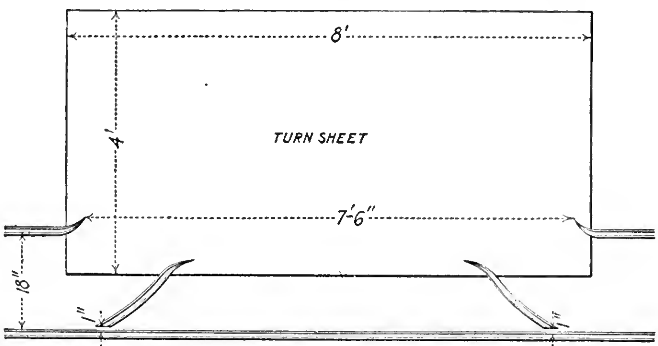


FIG. 377.—TURN SHEET AND SWITCH POINTS FOR TURNOUT.

full-length fish-plate, which holds the movable rail in its proper position when in use.

Turnout for Narrow Drift (By Albert G. Wolf).—In Fig. 377 is shown a cheap and simple turnout, designed by John B. Sommers, foreman of the Blue Jay mine of the Mason Valley Mines Co. One rail is unbroken while the other is left open for a space of 7 ft. 6 in. and the ends turned out for about 3 in. and pointed. Two pieces of rail about 2 ft. long, cut and bent, as shown, are placed at the proper distance from the points to give the gage. A clearance of 1 in. between the ends of these pieces and the through-rail is sufficient to allow the passage of the wheel flanges. An 8 × 4-ft. turn-sheet is placed as shown, giving ample room for a car to sidetrack. This turnout is in use in a narrow drift where the loaded and empty cars must pass each other in going to and from the shaft. Its great advantage is the elimination of switch points and the overturned loads due to them. The loaded car follows the through-rail, while the empty is easily side-tracked, and the flaring ends of the rails make it easy to run the car on again.

Cheap and Satisfactory Turntable (By L. O. Kellogg).—For transferring mine cars from track to track, underground or on the surface, either a system of switches or a large, well-leveled and well-backed slick

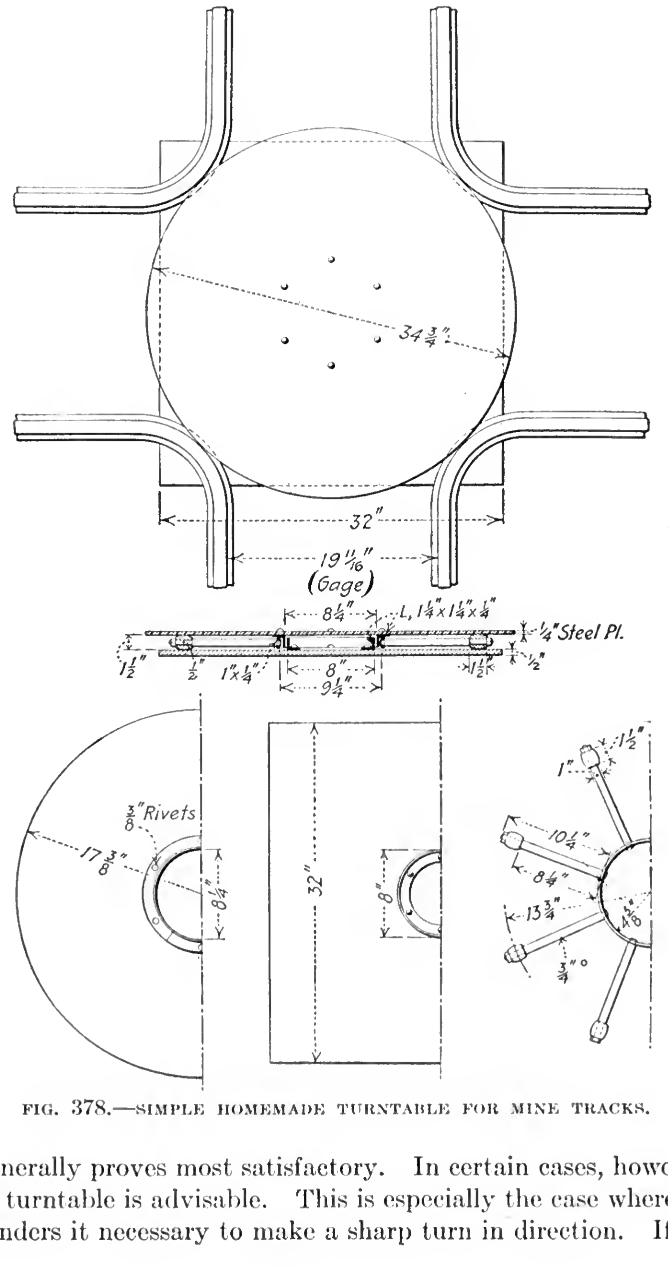


FIG. 378.—SIMPLE HOMEMADE TURNTABLE FOR MINE TRACKS.

sheet generally proves most satisfactory. In certain cases, however, the use of a turntable is advisable. This is especially the case where lack of room renders it necessary to make a sharp turn in direction. If a turn-

table is to be used, there is none better than that illustrated in Fig. 378. It is a design suggested at a Mexican mine by a mechanic engaged on a temporary erecting job. It proved an excellent device in practice and so far as known, has never been described in print. It consists of a fixed bottom-plate, a movable upper-plate and an intermediate spider carrying rollers. The relation of the three pieces is maintained by means of three collars which nest together. As shown, the table is applied to a track of 0.5-meter gage with 16-lb. rails; the dimensions are given in English units. The bottom plate is here shown as made of a ½-in. iron casting taken from the scrap heap. Other types of plates would answer, such as wooden backing covered with an iron sheet fastened with countersunk screws. The top plate is of ¼-in. steel. The spider has eight arms of ¾-in. soft steel turned down at each end. One end enters a hole in the ¼ × 1-in. collar and is riveted over. The other end carried a roller free to revolve on the turned portion and held in by a nut. The roller is of the shape shown, a prolate spheroid with truncated ends, to be exact. If the threads are bradded down a little, there is no trouble experienced from the nut's working off. The spider collar is welded so as to be continuous. The plate collars are of bent angles, riveted to the plates, no attempt being made to weld the ends. The diameter of the collars is such that the one on the upper plate fits loosely between that of the spider outside and that of the lower plate inside. Their height is such that the edges do not touch above or below in any case, the only contacts being those of the rollers. The car wheels come close enough to the line of the rollers so that for a 2400-lb. load with a ¼-in. steel plate, there was no dishing toward the center, so far as could be seen, which would make the collars bind. The tracks are laid as bends over the corners of the bottom plate and the thickness of the rollers is determined by the height of the rails, the top of the movable plate being at such a height as to take the bottom of the wheel flange as the tread leaves the rail. The edge of the revolving plate is made to come just below the inside edge of the rail head at the bend. The table is cheap to put up; it can be made at any mine shop that has a lathe. It is reliable and durable; it can never wobble, as often happens with tables revolving on a central axis; it is particularly easy to clean, since the top plate can be lifted off with one hand; and it is much simpler and less expensive than the rather elaborate tables offered by manufacturers.

XII

SAFETY AND SANITATION

Hoisting—Collar Protection—Haulage—Change Houses—Latrines, Etc. —First Aid—Miscellaneous

HOISTING

Safety Alarm for Slack Hoisting Rope.—Devices are used at the Harold mine on the Mesabi range which will automatically signal the engineer, should a hoisting rope become slack. They consist essentially of cradles of electric conductors stretched under the hoisting ropes and arranged to ring a bell in the hoist room when contact is made with a rope.

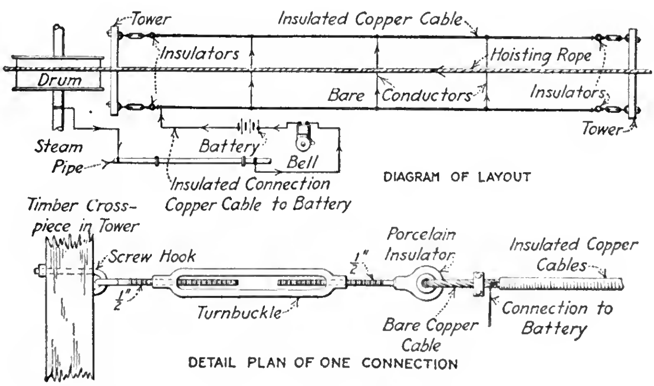


FIG. 379.—ELECTRIC SIGNAL ARRANGEMENT FOR HOISTING ROPE.

The accompanying drawing, Fig. 379, shows the arrangement diagrammatically and the detail of one of the four fastenings. As is general practice on the Mesabi, the hoist is some distance from the shaft and the ropes are supported over this interval by pulleys on wood towers. Two insulated copper cables of twisted wires are stretched between two towers a few feet under each rope and about 18 in. apart. The ends of the cables are held by turnbuckles and porcelain insulators. Several crosspieces of naked copper wire are connected to the cables and one cable is connected through a battery to the hoist-room bell and grounded to the main steam pipe. The slack rope falling on the naked cross-wires completes the circuit through the hoist and rings the bell. The hoist room being

inclosed, the engineer cannot see what is going on at the headframe. Slack rope indicates trouble from two possible sources. The skip just dumped may get hung up, especially during freezing weather when ice and snow will often jam it; then when the engineer starts the next trip, the skip cannot pull out the rope, which becomes slack and rings the warning. Or the skip may have been hoisted too high, in which case the bottom skip lands and fails to pull out its rope and the slack rope rings the bell again. In either case the engineer will stop his engine and investigate.

Electric Indicator for Hoist Reverse.—The Rogers-Brown Ore Co., Crosby, Minn., is using an electric device to warn the hoisting engineers

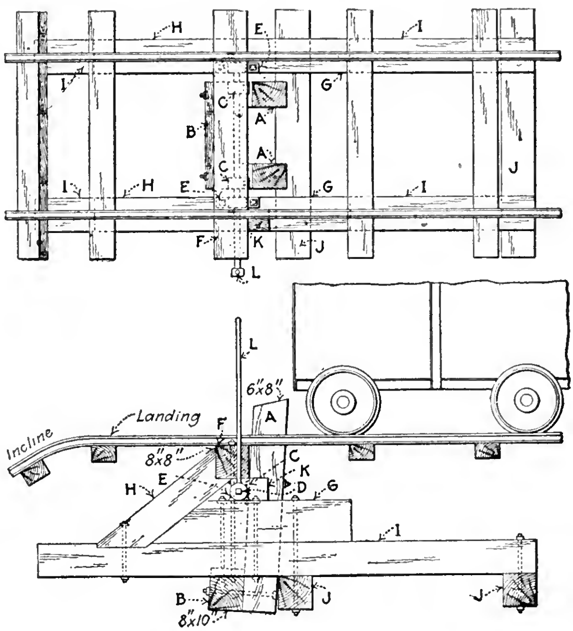


FIG. 380.—AUTOMATIC SAFETY STOP FOR TOP OF INCLINE.

that the skip is nearing the collar or has passed it. In the words of its inventor, J. P. Wallheus, master mechanic for the mining company, the device is intended "to keep the engineer's mind on the reverse lever and to warn him when the skips are near the dump." A 60-c.p. electric lamp is mounted on a board in front of the engineer and is painted red. This lamp, reverse lever and indicator are connected to an electric circuit. The circuit is open until the skips are about 30 ft. from the dump, at which point the circuit is closed by an automatic switch attached to the indicator. This turns on the red light, showing the engineer that the skip is near the dump. The red light remains on until the engine is

reversed, when the circuit is opened and the red light disappears. For the cage hoist, which does not run in balance, the automatic drop switch is on the indicator. This switch will drop only when the cage is hoisted a little above the collar of the shaft, and at this point the red light will flash on, indicating to the engineer that he has hoisted too high or that he is going in the wrong direction, as the case may be.

Safety Block for Incline Top.—To prevent the accidental return of a car or skip that has been hoisted to the top of an incline and detached, the device shown in Fig. 380 is recommended by William W. Jones, state mine inspector, Albany, N. Y. It consists of two almost upright timbers *A*, fastened to the square shaft *D* by the straps *C*. The shaft is turned in two places to fit the boxes *E*. The bottoms of the timbers *A* are bolted to a transverse piece *B*. The hoisted car hits the timbers *A*, which revolve with the shaft, permitting the car to pass. The weight of *B* then brings them to the upright position and the crosspiece *F* prevents their swinging in the other direction. To release the car, the lever, which is attached to a square portion of the shaft, is used to force the timbers down below the level of the axles. The bill of material for installing the device on an incline with a track of 3-ft. gage is given in the table.

Bill of Materials for Safety Block; Track 3-ft. Gage

	Timber	Iron
A	2 pieces 6 × 8 in. by 4 ft. 4 in.	4 bolts ½ × 20 in.
B	1 piece 8 × 10 in. by 2 ft. 2 in.	2 bolts ⅝ × 14 in.
F	1 piece 8 × 8 in. by 5 ft.	2 bolts ⅝ × 17 in.
G	2 pieces 8 × 10 in. by 4 ft.	2 bolts ¾ × 28 in.
H	2 pieces 8 × 8 in. by 3 ft. 2 in.	2 bolts ¾ × 34 in.
I	2 pieces 8 × 8 in. by 10 ft.	4 bolts ½ × 10 in.
J	2 pieces 8 × 8 in. by 5 ft.	2 bolts ½ × 18 in.
K	1 piece 5 × 6 in. by 1 ft. 1 in.	1 shaft 115⁄16 × 115⁄16 in. by 4 ft. 4 in.
		1 piece ⅜ × 1½ in. by 5 ft.
		2 clamps for shaft
		2 boxes 2½ × 8 × 2 in. bore.

Car Catch at Incline Top (*Coal Age*).—A safety stop similar to that recommended by W. W. Jones, is illustrated in Fig. 381. The arrangement needs little description. It consists of a square bar, thrust through a 7- or 8-ft. piece of scrap rail at a point above the center of gravity of the latter, and with its ends rounded so as to turn under two clamps set on timbers, as shown. If it is desired to permit the return of the cars down the same incline, a lever can be attached to the end of the bar.

Light Safety Crosshead (By Roy Marcellus).—The drawing, Fig. 382, shows a light crosshead used extensively in the Coeur d'Alene district, Idaho. It conforms with the state laws regarding safety attachments for use in sinking. The spring *A* is compressed when the load is freely

suspended; when tension is removed, as by the ropes breaking, the spring extends and pulls the dogs *E* into the face of the guides. The drawbar *B*

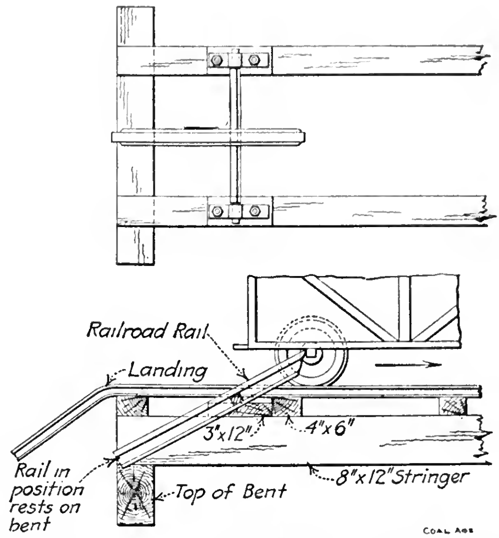


FIG. 381.—SAFETY CAR CATCH MADE OF BAR AND RAIL.

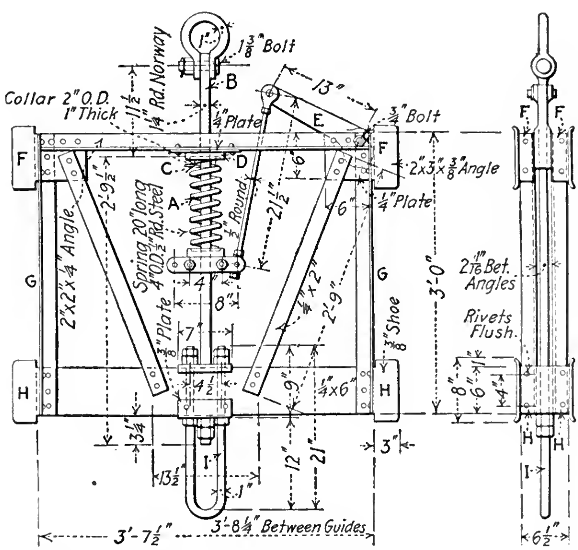


FIG. 382.—STEEL BUCKET-CROSSHEAD WITH SAFETY DOGS.

has a collar *C* welded on it to rest against the spring cup *D*; this, with the nuts at the bottom, carries the load. The dogs are made entirely of steel and have teeth forged across their 2-in. faces. The upper shoes *F*

are made of angle iron riveted to the vertical angles *G*, having clearance in the center for the operation of the dogs; the lower shoes *H* are made of plate, as usual. The bucket is attached by a chain to the U-bolt *I*. The crosshead can be built as light as 300 lb.

[It should be noted that the crosshead must ride with the bucket, since the rope does not pass through, as in the ordinary type of wooden cross-

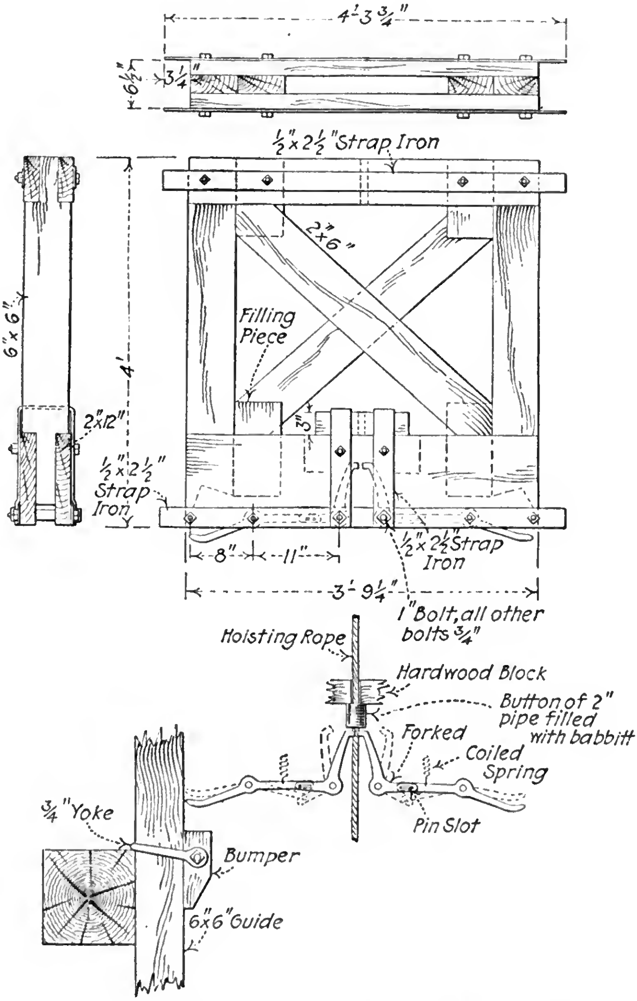


FIG. 383.—SAFETY CROSSHEAD WITH AUTOMATIC RELEASE.

head. This means that the bucket must be swung from a long chain when sinking, in order to extend to the shaft bottom when the crosshead is stopped at the end of the guides. A long connection such as this means considerable extra height in the headframe to permit dumping.—EDITOR.]

Homemade Safety Crosshead (By Lowe Whiting).—The accompanying drawing, Fig. 383, illustrates a homemade crosshead that was used in the Iron River district of Michigan, during the sinking of a small shaft to a depth of 450 ft. A "button," made of a piece of 2-in. pipe, 3 in. long, was fastened to the rope. This was slipped over the rope where the strands had been slightly separated, a small pin put through two holes in the pipe and through the loosened strands, and the whole filled with babbitt. In descending, the crosshead strikes the bumpers at the bottom of the timbering, releasing the claws, allowing the "button" to pass between them and the bucket to continue to the bottom of the shaft. On hoisting, the button strikes the 3-in. piece of hardwood, lifting the crosshead, which allows the coiled springs to again bring the claws into action.

Bucket Dump-hook.—In the Joplin district practically all the ore is hoisted in buckets. The hoisting engine is set in a four-post derrick frame next to the shaft, and the hoistman hooks the tail, or dumping,

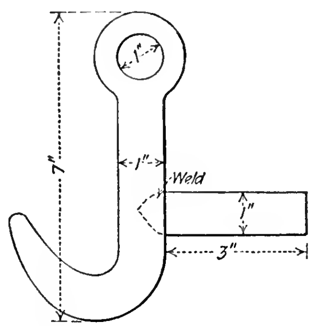


FIG. 384.—DUMP-HOOK USED ON JOPLIN BUCKETS.

rope into the ring in the bottom of the bucket, and slacking off on his friction brake, dumps the bucket. In order to allow the hoistman to grab this hook and at the same time to swing over the door that closes the shaft, this bucket hook is made with a handle sticking out behind, as shown in Fig. 384. This handle should start out from the shank of the hook about level with the tip. This makes the missing of a ring less likely to occur, as the hooking is then a punching action directed so as just to miss the ring in the bucket. The advantage of the handle, no matter where it is put, is that the point of the hook is always at a constant distance above the hand, and as all the fingers hold the handle and none is used around the shank of the hook there is no danger of mashing the hand.

Safety Bucket Hooks (By F. C. Rork).—The accompanying drawing, Fig. 385, shows two styles of bucket hooks which are quite common in Canadian mines. They comply with the Ontario laws, which specify that no open hook shall be used in hoisting or lowering. The hooks can

be made by any blacksmith from material which is usually kept in stock at a mine and combine safety with ease and speed of operation.

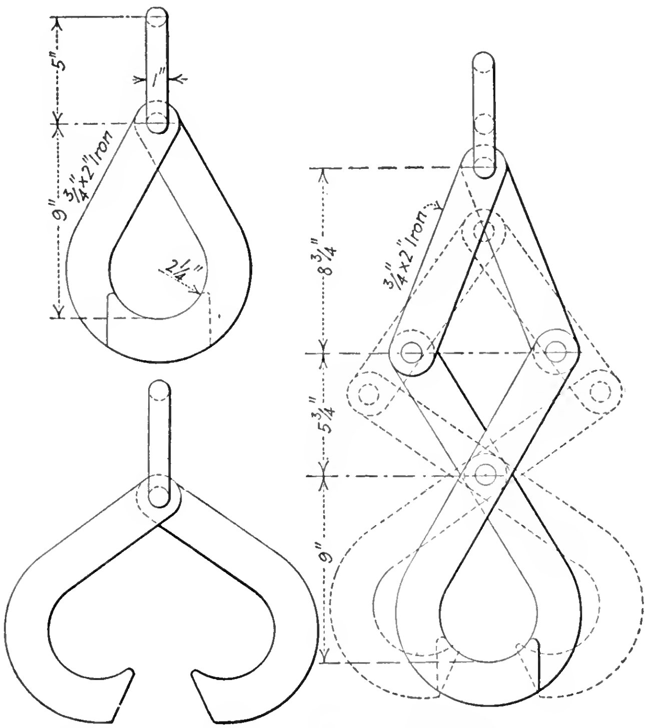


FIG. 385.—BUCKET HOOKS COMPLYING WITH ONTARIO LAW.

Sliding Chain Gates on Cage (By F. H. Armstrong).—For a safe yet easily handled gate for a cage, bars of $\frac{3}{4}$-in. pipe fastened to two chains are used, as shown in Fig. 386. The chains run over pulleys set at the

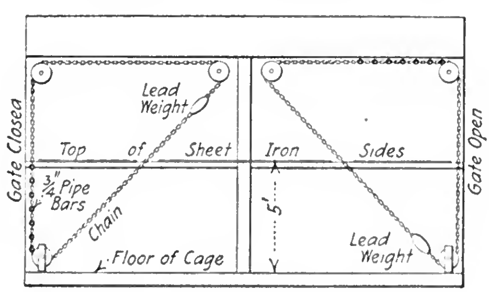


FIG. 386.—CAGE GATES OF CHAIN AND PIPE.

corners of a triangle. On the hypotenuse of the triangle is a lead counterbalance to offset the weight of the bars. When the gate is open, the bars

are on the horizontal part of the chain along the roof of the cage and the weight is near the bottom of the cage. When the bars are down—gate closed—the weight is near the top of the cage. This appliance is simple, reliable and easily operated.

COLLAR PROTECTION

Lifting Guards for Shaft Collar.—In the lead district of southeastern Missouri, the ore lies directly on top of a water-bearing sandstone in which deep sumps are not desirable owing to the difficulty in handling the water. Due to the shallow sumps it is not possible to use cages having more than one deck. Thus in order to handle the production of the mine, relatively high hoisting speed must be used and all unnecessary delays

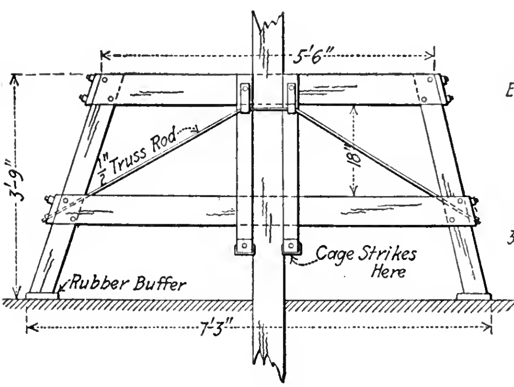


FIG. 387.—LIFTING RAILING FOR SHAFT.

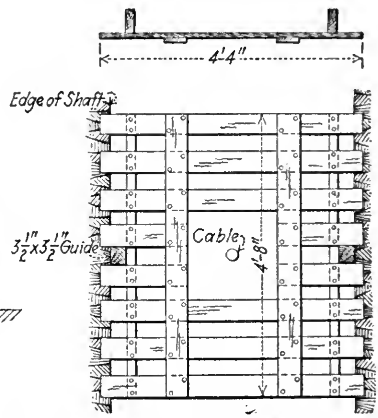


FIG. 388.—PLATFORM SHAFT COVER.

in putting the cars on and off at the surface avoided. This fact is made clear when it is known that as much as 400 to 500 tons are regularly hoisted in eight hours from a depth of approximately 450 ft. on a one-deck cage using 1-ton cars, and that often as much as 800 tons are handled at some shafts using 1½-ton cars. This necessity for cutting down delays has developed several types of automatic guards for shaft collars which are operated by the cage itself. Such a guard, whether it be a railing or a platform, must be made as light as consistent with strength in order to diminish the shock when it is picked up by the cage. A common form of platform consists of 1 × 4-in. planks, made up as shown in Fig. 388. At the Desloge Consolidated shafts, a lifting guard railing or frame is used; it is no heavier than the platform and is regarded as much safer. The construction of this frame is shown in Fig. 387. The lower braces are notched clear into the corner posts, while the upper ones are notched

in only about half way. A ½-in. steel truss rod is used to distribute properly the stress due to the sudden lifting of the frame. This rod passes through loops in the straps by which it is fastened to the top cross-pieces of the framework. The corner posts are placed at an angle to give added strength as well as to decrease the weight of the top of the structure. Rubber bumpers are used to take up the jar of the frame when it drops back upon the landing floor.

Safety Bonnet for Shaft Opening (By E. H. Edyvean).—Fig. 389 shows a safety guard or bonnet protecting the cage compartment of the

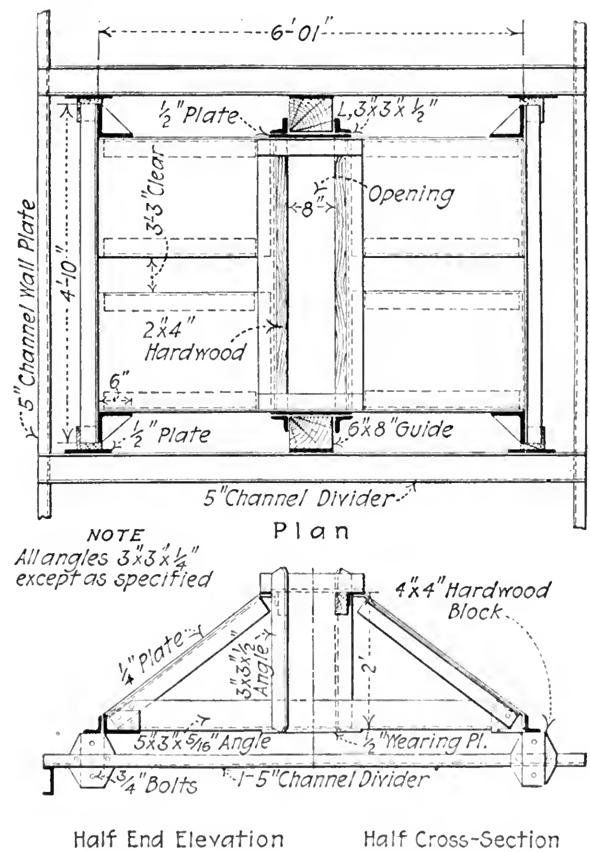


FIG. 389.—SHAFT COVER OPERATED BY CAGE.

shaft at the Bristol mine, Crystal Falls, Mich. It is in use at a point in the shaft house, some distance above the surface, where the waste rock is landed after hoisting in the cage. The bonnet is constructed of steel with a hardwood lining for the top opening. The 3 × 3 × ¼-in. end angles are extended 6 in. outside the body of the bonnet and serve

as the means of support. These ends are reinforced with corner brackets as shown in the drawing. The sloping sides are made of ¼-in. steel plate, reinforced with 3 × 3 × ¼-in. angles. The opening between the plates gives plenty of room for the lever operating the safety catches on the cage. The shoes are of the same gage and move on the same guide as those on the cage. The wood lining in the top opening protects the top frame from the hoisting rope. The bonnet is held over the shaft by four supports, one on each corner. These consist of ½-in. steel plates securely riveted to the dividers, and of hardwood blocks bolted to the plates; the blocks are easy to replace in case they are broken. The yoke of the ascending cage catches the bonnet, raises it and supports it while the cage is at the landing. When the cage descends, the bonnet drops on its supports and closes the shaft opening almost entirely, thus serving as a simple and effective safety device.

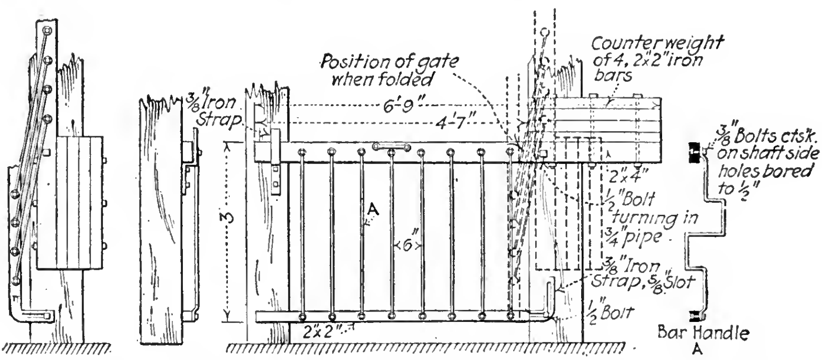


FIG. 390.—SHAFT-COLLAR GATE OF WOOD AND IRON BARS, ARRANGED TO FOLD.

Folding Gate across Shaft (By W. H. Jobe).—The accompanying illustration, Fig. 390, represents a shaft gate designed by Capt. Edward Jacka, of the Armenia mine, Crystal Falls, Mich. It consists of two horizontal wooden bars, the top one 2 × 4 in., the lower 2 × 2 in. These are connected by eight vertical iron bars bolted loosely at the top and bottom. The upper horizontal bar pivots on a ½-in. bolt, turning in a ¾-in. pipe in the shaft timber. The lower horizontal bar ends in an iron strap turned at right angles, ⅜ in. thick with a ⅝-in. slot along its middle. A ½-in. bolt through the slot holds the strap to the shaft timbers, but permits it to slide. Thus, as the gate is raised, it folds up as shown. A counterweight of four 2 × 2-in. iron bars is bolted to the upper horizontal bar. One of the vertical bars has two offsets as shown, and serves as a handle.

Hinged Shaft Bar.—The 2 × 6-in. wooden bar across the man-cage compartment at the Zenith mine, Ely, Minn., is hinged in the middle.

This permits it to be lifted and lowered somewhat more quickly, makes it less likely to fall accidentally when standing open and requires less headroom to accommodate it. The manner in which it is pivoted and supported is shown in Fig. 391. The hinge is bolted to the lower side, so that by placing the hand under the bar at the point A, the bar can be lifted with one motion. The hinge is made in the shop and is unusually stout. The ends of the bar are run through vertically with ½-in. bolts

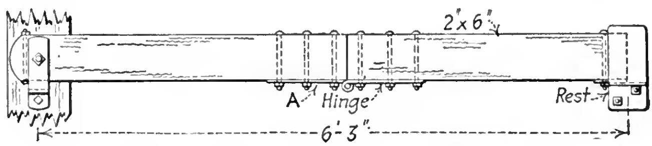


FIG. 391.—BAR ACROSS MANWAY COMPARTMENT.

as a precaution against splitting and the loose end has an iron wearing plate underneath.

Swinging Shaft Gate of Iron (By W. H. Jobe).—A shaft gate used by the Verona Mining Co. of Michigan is shown in Fig. 392. It consists of a frame of pipe and iron fence fittings braced with strap iron. The gate is hung at a slight inclination, so as to swing shut of its own weight. An iron plate, ⅛ × 6 in., is fastened to straps from the gate at the bottom and serves as a "toe board."

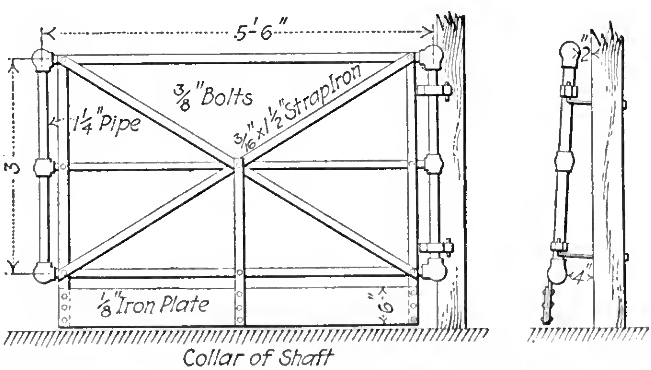


FIG. 392.—GATE OF PIPE AND STRAP IRON, SWINGING SHUT BY GRAVITY.

Handy Gate Latch.—The gate latch illustrated in Fig. 393 is one used at the Bennett mine, Keewatin, Minn. While in this particular instance it was applied to the gate into the manway, through the fence around the shaft collar, it is suitable for any gate and is extremely neat, reliable and convenient. It consists of a horizontal finger attached to the gate and a ring held loose in a double-strap hanger fastened to a corner post of the fence. The 2½-in. ring rests on a ¼-in. rivet between the two straps.

In closing the gate the finger strikes the ring, which is so positioned as to move freely inward and upward and allow the finger to pass. The gate cannot open by itself, as the finger strikes above the center of the ring so that the latter cannot lift; nor can it swing outward, since it strikes the closed end of the double strap; the latch is released by lifting the ring with the hand.

Protective Combing for Manway Top.—Cribbed manways are raised through vertically from level to level in the mines at Ely, Minn., operated by the Oliver company. The tops of these are covered with planks spiked to the cribbing and a rather small opening is left around the ladder. In order that material may not accidentally be kicked down the manway by anybody approaching its top, a low railing or combing is nailed around the opening. This is of 2 × 6-in. material, constructed in the manner shown

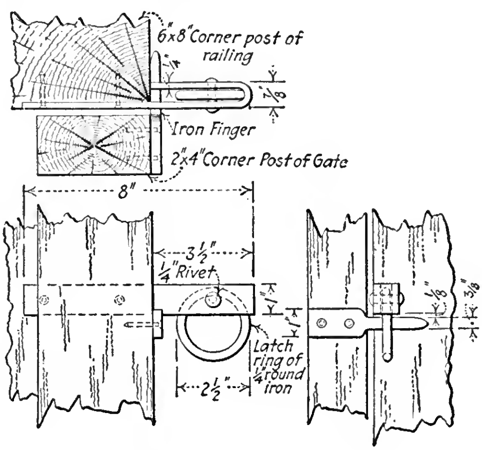


FIG. 393.—RING AND FINGER LATCH ON SHAFT GATE.

in Fig. 394. It was found that the opening at the manway top, while large enough to permit the passage of a man, made handling of the Draeger resuscitation apparatus difficult, if circumstances should compel it to be taken into the stopes. To get more room, therefore, another board on the side of the opening opposite the ladder was cut away and fitted with hinges, so that it could be thrown back when necessary, although usually kept closed. The dimensions and arrangement of the opening, combing, etc., are somewhat different in different manways, but the illustration shows a typical case.

Safety Door for Chute Top (By H. H. Hodgkinson).—Underground trammers will persist in leaving open the doors over ore chutes, exposing others to the danger of falling in, if the chutes are in the main traveling roads. Such chutes should always be provided with doors, and they

should be closed when not in use. A piece of ¼-in. plate 22 × 42 in. will make a handy door for a chute between the rails when a 2-ft. gage track

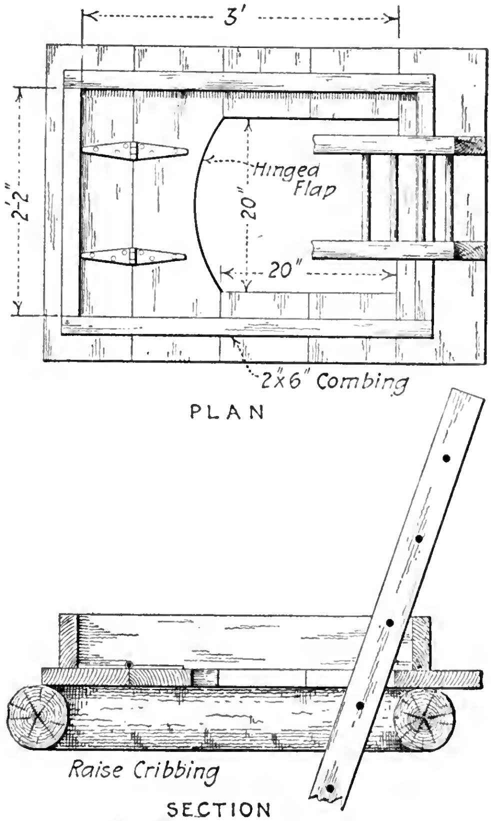


FIG. 394.—MANNER OF PROTECTING RAISE TOPS.

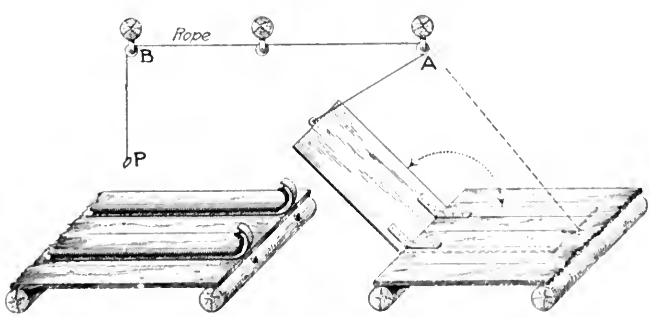


FIG. 395.—CHUTE COVER OPENED AND CLOSED WITH ONE ROPE.

is used. A piece of ⅜-in. hemp rope about 18 ft. long and two small pulleys will not only save time and make it easier for the men pushing the

cars to open and close the doors, but will help them to remember that they must do it. The arrangement is shown in Fig. 395. By pulling the rope at P with a quick jerk, the door will swing all the way back and rest on the platform, and when in the latter position another jerk at the rope will close it. The pulley A is directly over the hinges of the door and is fastened to the cap of a timber. The pulley B is also fastened to a cap about 12 ft. from A directly over the center of the track. The rope at P hangs down in the center of the track and is low enough to touch the trammers on the shoulder. The rope should always be extended far enough along the drift to enable the trammers to open the door just before they reach the dump, thus avoiding delay.

HAULAGE

Automatic Gong for Underground Motor.—The excellent safety device illustrated in Figs. 396 and 397 consists of a gong so attached to the wheel of an underground electric locomotive as to ring whenever the wheel revolves, and thus automatically signal the motor's approach to anybody in the haulageway. The gong in the form illustrated was developed at the Oliver mines at Ely on the Vermilion iron range after a good deal of experimenting. The first type of gong used for the purpose involved a spring and trigger arrangement which required constant attention and repair. In the effort to avoid this difficulty the scheme to be described was hit upon.

The gong is attached to the center of the wheel on the outside, as shown in Fig. 397. The details of construction are exhibited in Fig. 396. A 1¼-in. piece of round iron is turned down at each end to form two shoulders and one turned portion is threaded. This is slipped through the hole in the center of the 10-in. gong and fastened to it with a nut. The other end is riveted to an iron plate, 10 in. in diameter to correspond to the gong and about ⅛ in. thick. The length of the pin is such that the plate clears the gong about ⅛ in. A piece of flat iron is wrapped partly around the pin and its ends shaped to correspond roughly to the inside of the gong. This is riveted through the pin and the ends are also riveted together, forming a paddle. An iron or steel ball about an inch in diameter completes the device. The plate is fastened to the wheel with two bolts, not shown in the illustration. The gong, plate, pin and paddle revolve with the wheel, the ball being the only loose part. The paddle catches the ball and lifts it to a point near the top, when it rolls down the paddle and drops from the pin, striking the lower edge of the gong and ringing it. The device seems perfect except in one respect: At 55 to 60 r.p.m., centrifugal force is sufficient to overcome the force of gravity and the gong revolving at that speed will not ring. With the 28-in. wheel used on

the motor, this corresponds to about 5 miles per hour. The company's mechanical department has expended considerable ingenuity in the attempt to overcome this defect and has tried various devices. One of these worked successfully at high speeds, but failed at low speeds and furthermore was so complicated as to destroy the chief merit of the device, its simplicity and strength. Five miles per hour, it should be noted, is fast enough for underground tramming under most conditions. The design shown is that used where the wheels are outside the motor casing. When the wheels are inside the casing is pierced at a point

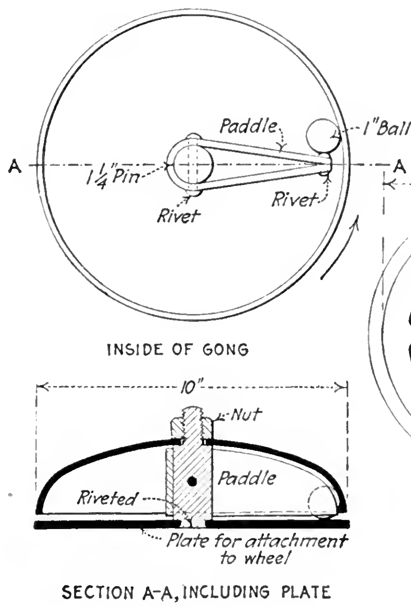


FIG. 396.—DETAILS OF GONG.

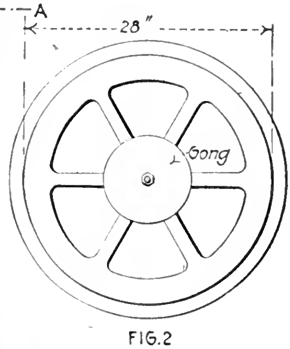


FIG. 397.—GONG ATTACHED TO WHEEL.

corresponding to the center of the wheel and the pin of the gong extended through the hole and into the wheel center as a bolt, the device thus revolving on the outside of the casing.

Hanging Troughs for Protecting Trolley Wires (By Claude T. Rice).—To guard against the danger of electric shock from underground trolley wires some form of protection is necessary. This usually takes the form of a trough, the sides of which extend down below the wire and keep men from hitting the wire with their bodies or with tools they may be carrying on their backs. At chutes the danger is especially great, but it costs little more and makes the workings far safer, to extend the trough protection throughout the mine.

Different methods for supporting the protecting trough have been

suggested. Probably the cheapest is a modification of the system used to carry the insulators at the Mascotte tunnel, Bingham, Utah. Here stoping drills put two inclined holes into the adit roof, 12 in. deep and 2½ in. in diameter; 4 × 4-in. pegs sharpened to fit are driven into the holes. As these two pegs slope away from each other, the trolley trough would be securely held to the roof if it were nailed directly to their squared heads, as shown in Fig. 398, 1. This method is practicable when the roof is of uniform height, so that long plugs are not necessary to keep the trolley board approximately level.

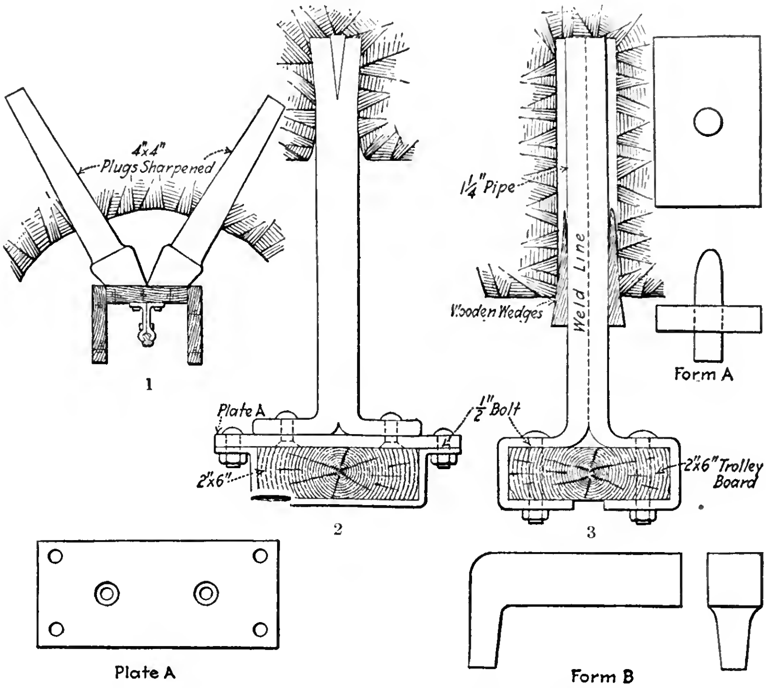


FIG. 398.—FORMS OF TROLLEY TROUGH HANGERS.

A better method, adaptable to all roofs, is that used at some of the mines of the Cleveland-Cliffs Iron Co. in the Lake Superior district. Here steel pins are fastened in vertical holes by a wedge driven into the split inner end of the pin, as seen in 2. The outer end of the pin is also split and bent out to form wings. These wings are flattened and to them is riveted a ½-in. plate carrying four holes through which ½-in. carriage bolts pass and support the straps carrying the protecting trough. The plate is made wide enough for four bolts, to allow splicing the troughs at the pins. It is just as well, however, to splice the troughs with top or side cleats and so have the wings carry the straps directly without the

intervening plate. This makes a strong hanger where the trough has to be carried at some distance from the roof, and it is a good way to use up old drill steel. At the Cliff shaft of the Cleveland-Cliffs company a somewhat less expensive hanger was devised by Lucien Eaton, superintendent of the Ishpeming mines of the company. Pieces of 1¼-in. pipe are fastened by wooden wedges in holes in the roof 10 in. deep as shown in 3. The outer end of the pipe is split, one cut following the weld. This end is then spread out and the wings flattened on forms. The first form *A* consists of a flat piece of iron 1 in. thick, 4 in. wide and 6½ in. long. A pin at the center of the form projects both top and bottom; the top end enters and centers the pipe, while the bottom end is shaped to enter the hardie hole of the anvil and hold the form. The wings of the pipe having been bent down and flattened are bent at right angles over the ends of the form. The pipe with the wings still hot is now completed on the second form *B*. This is merely a 2 × 2-in. bar of iron with its end bent down and shaped to enter the hardie hole. On this form another right-angle bend, back toward the center of the hanger, is made in each wing. In this way a 6½ × 2-in. yoke is formed to receive the top board of the trolley trough. The yoke is bored for two ½-in. bolts, one on each side, by which the 2 × 6-in. trolley boards are securely fastened. With this system the trolley boards are spliced by the side boards of the trough or by a top cleat. By the use of forms a uniform size and shape of hanger is insured and much time is saved in the blacksmith shop. A good blacksmith can easily make 10 hangers per hour. As all scrap pipe and short lengths that are in a fair condition can be utilized, the hangers are relatively cheap and, unless unduly long, are quite strong enough. Measurements are taken at the different points along the drift and the hangers, properly numbered, are made and sent down ready to be put 10 in. into the plug holes and yet bring the trolley board level, so that the trolley will run at approximately the same height throughout the mine, which is desirable for the best operation. Aside from the advantage derived from the use of the protecting troughs from the standpoint of safety the top board of the trough acts as a guide to the trolley pole should it jump the wire, and thus saves it from injury. The Cleveland-Cliffs company, while it does not use a trough along the entire line, does use a top board, to carry the insulators. Troughs are used only where there is much travel, and near chutes.

V-shaped Trough for Trolley Wire (By Allen H. Foster).—Supplementary to the article by Claude T. Rice, the accompanying illustration, Fig. 399, shows a trough which is useful where there is little headroom available. The stringer boards *A* are 1- or 2-in. planks, depending on the span between supporting plugs. Blocks *B* are placed at the points where trolley ears are located, and wherever it is thought desirable to stiffen the

trough. Under comparatively dry conditions the petticoat insulator *S* can be omitted and the ear attached directly to the block *B*. In this case the blocks should be treated with an insulating compound.

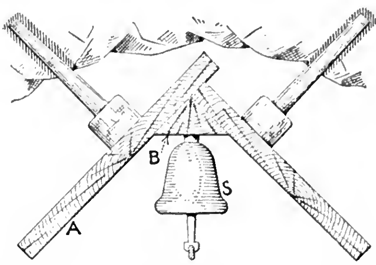


FIG. 399.—INVERTED-V TROLLEY-WIRE PROTECTING TROUGH.

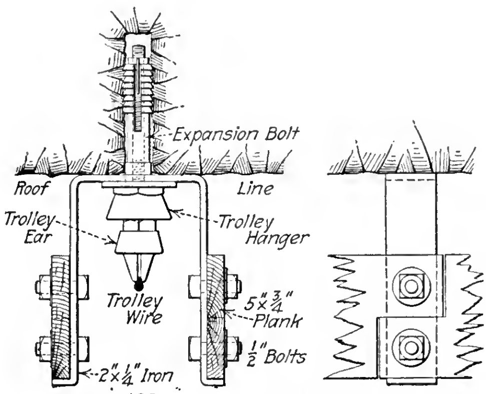


FIG. 400.—DOUBLE HANGER SUSPENDED FROM ROOF.

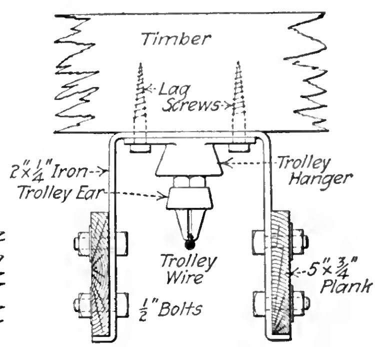


FIG. 401.—DOUBLE HANGER SUSPENDED FROM TIMBERS.

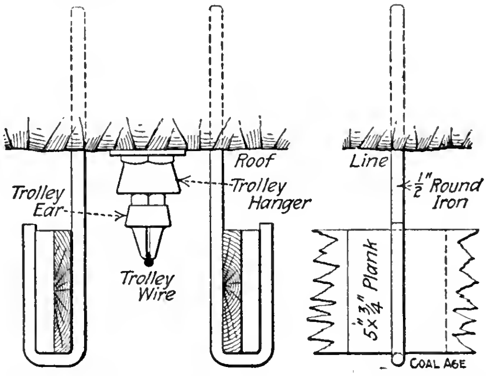


FIG. 402.—SINGLE INDEPENDENT HANGERS DRIVEN INTO ROOF.

Trolley-wire Troughs for Uniform Height of Back (*Coal Age*).—A method of suspending troughs for trolley wires, where it is not necessary

to allow for serious inequalities in the height of the back, is shown in Figs. 400, 401 and 402. An iron strap is held by each trolley hanger, and boards are bolted to these. If cut to length, they can be notched and joined at the hangers, as shown in Fig. 400. They should be wide enough and placed low enough to cover the point of greatest sag in the wire. Methods of attachment to rock roofs and to timbers are shown in Figs. 400 and 401, respectively. If the trolley hangers are spaced more than 15 ft. apart, intermediate straps should be provided for the boards. In case it is desirable to carry the protection independently of the trolley-wire supports, the method shown in Fig. 402 is available. In this case the boards are merely lapped in the bracket and no bolting is necessary.

Trolley-wire Protection of Round Lagging (By W. H. Jobe).—The accompanying illustration, Fig. 403, represents a method of protecting underground trolley wires, devised by Capt. E. Carlson. of the Bristol

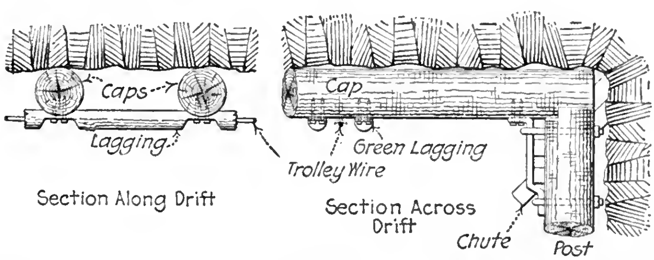


FIG. 403.—POLE GUARDS TO PREVENT CONTACT WITH TROLLEY WIRE.

mine, near Crystal Falls, Mich. Lengths of round green tamarack lagging are notched and spiked to the drift caps on both sides of the trolley wire. This material is readily had and is cheap. It is more easily placed than the ordinary inverted trough and is much stronger and capable of resisting the shock of blasting in the chutes, which is frequently necessary.

Safety Hand Grip for Car (By H. H. Hodgkinson).—When ore cars are heaped full, the trammers often receive badly bruised hands from the chunks on the tops striking chutes and rolling down, inasmuch as the men persist in gripping the angle iron around the rim of the car body. It is sometimes necessary, also, to hold back the car, in order to stop it or to prevent it from going down-grade too fast. There is no suitable place on the car for holding, and the angle-iron rim gives a poor grip. The convenience illustrated in Fig. 404 consists of a piece of 1¼-in. pipe threaded at each end and attached to the car by means of two 4 × 3 ×¼-in. angle irons, which have each a 1⅝-in. hole bored to fit the outside of the pipe. These two angle irons are riveted fast to the body of the car in such a position as to bring the grip just under the angle-iron rim, which will then act as a guard to the hands

of the trammers. A 1¼-in. pipe coupling is cut in half to take the place of a nut, half being put on each end of the pipe or grip.

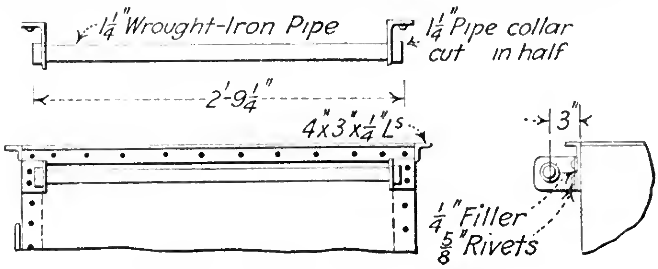


FIG. 404.—A PIPE-BAR GRIP FOR MINE CARS.

Shoeing Mean Mules.—Trouble is often experienced in shoeing mules, especially underground. In Fig. 405 are shown the features of a shoeing

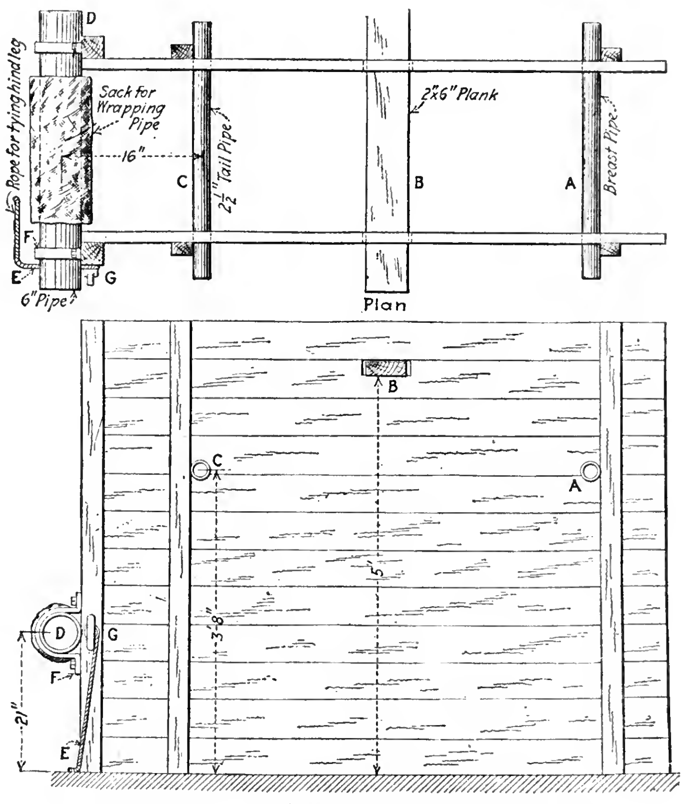


FIG. 405.—STALL FOR SHOEING REFRACTORY MULES.

stall at one of the shafts of the St. Louis Smelting & Refining Co., in the Flat River district of southeastern Missouri. This stall is made narrow

so that the mule has not much lateral leeway. There is a pipe *A* at the manger or head end of the stall against which the breast of the mule comes. This is put in after the mule has been forced back against the other pipes and the board that are used to restrain him. By this breast pipe he is kept from moving forward; by the board *B* that is put over his back he is kept from rising to kick; by the pipe *C*, which is put in at the same height as the breast pipe, he is kept from backing out of the stall; and by the pipe *D*, which is wrapped with old gunny sacks to prevent him from injuring himself, he is prevented from kicking. The foot that is to be shod is raised up over the pipe *D*, which is held in iron straps *F*, bolted to the back posts of the stall, and then is tied down by the rope *E* which is fastened to the cleat *G*. By this rope the mule is prevented from jerking his foot back out of the shoer's grasp. The mule can be turned around and his front feet also shod in this stall if he causes trouble by not standing still or by trying to strike the man. Some mules give trouble by laying their weight on the man. This can be remedied by rigging up a swing and putting a band or surcingle around the mule and swinging him so as to take his weight off the shoer.

CHANGE HOUSES

New Type of Copper Queen Change House.—Prior to 1913, the change houses or drys for miners of the Copper Queen Consolidated Mining Co. in Bisbee, Ariz., were all of one type, similar in equipment and construction to others in the Southwest. They were usually two-story wooden buildings, with corrugated-iron roofs and walls. They had wash rooms and shower baths, and lockers for clothing. The lockers were ventilated and had steam pipes beneath. Each change house had its own heating plant for warming the building and heating water. Exhaust steam was not available, as all power was generated at a central station and distributed by electricity or compressed air. When new buildings were designed, an attempt was made to improve on the older construction, particularly in the directions of sanitation and reduction of maintenance costs. In the old buildings, the wooden floors wore rapidly, and gradually became saturated with wash water, etc. Although the lockers were ventilated at top and bottom, wet clothes were packed too closely to allow free circulation of air. It was necessary to keep the rooms uncomfortably hot in order that the clothes might be dried. The Uncle Sam change room was built in the winter of 1912–13. It was followed by one at the West Atlanta shaft and later by a larger building at the Sacramento shaft. These are all single-story buildings, with cement floors. Small lockers are used for such articles as might be lost or stolen, but all clothing is hung from the ceiling. The wet clothes are thus spread out to

favor drying by free ventilation, rather than by the heat of steam pipes. Each change room has a washroom and shower baths, accommodations for foreman, bosses and timekeeper, and a small storeroom.

For one end of the Sacramento building a cut in the ground was necessary, while the other was built on fill. The building was surrounded by shallow concrete foundation walls and the fill was settled by water. The cement floor was laid on the ground, with the exception of the wash room, which was supported by iron posts and laid on Berger plate. The skeleton of the building is of timber, plastered within and without on Hy-rib. Fig. 406 is a plan of the change house. The heating plant, separated

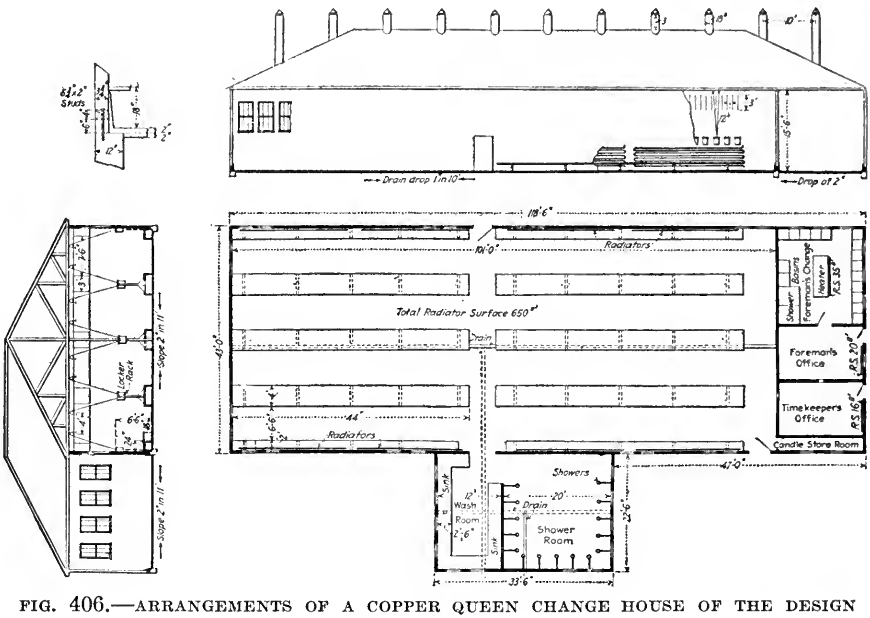


FIG. 406.—ARRANGEMENTS OF A COPPER QUEEN CHANGE HOUSE OF THE DESIGN RECENTLY ADOPTED.

from the change room to cut down the fire risk, is situated below the building, so that condensed water drains back to the boiler. The heater is an "Ideal" steam boiler of the American Radiator Co., 36-9 section. For heating the main room 650 sq. ft. of radiating surface is provided, and 70 more for the office and the bosses' change room. The steam pipes are laid around the walls above the benches, and protection from burns is given by wooden gratings. The floor of the building is clear for washing with a hose; nothing but the iron supports of the benches interfere.

In Arizona, during a large part of the year, the air is so dry that clothes may be dried without heat if there is a free circulation of air. To furnish this, ventilators are spaced at 10-ft. intervals in a row down the center of the building. The room is exceedingly wide, and it may be desirable to

provide additional ventilators. It would be even better to leave out the flat ceiling, which would allow the air to pass still more freely through the clothes. Thin sheets of galvanized iron separate the surface and mining clothes, but they appear to be unnecessary. Miners work eight hours in the Warren district, and clothes have at least 14 hr. in which to dry. As soon as the morning shift has changed, the fire is banked and all windows opened. Little fire is necessary until an hour or an hour and a half before the shift comes up, when the windows are closed and the fire freshened to warm the room and heat water for bathing. The men on the night shift go down an hour and a half after the day shift comes up, and the building is still warm for them while they change their clothes. The fire is again allowed to die down and is started up to warm the room for men coming off shift at 1:30 a.m. and for the morning shift which goes on at 7:30.

The cost of the building was as follows: Excavation, $303; concrete foundation walls, and floor, $1589; building, $10,490; heating plant, etc., $3056; total, $15,438. The building is large enough for 500 men, but only 383 lockers are provided. The cost of extra lockers would be about $250 more, making the cost per man $31.37.

The buildings have not been constructed long enough to give a fair average for cost of maintenance. The expense for fuel is more uniform, and a shorter time is sufficient to give dependable costs. It was found that in the new change rooms the cost of fuel per man for a period of nine months ended not long ago was only 72 per cent. of that of the older type. None of the new change rooms is used to its full capacity, while the older buildings are crowded. The saving in fuel per man would be even greater if they also were filled. It may be safely assumed that these buildings will cost less for repairs than the earlier type, and there will be an important saving in fuel. There is also a great improvement in sanitation. The usual rather unpleasant smell of drying clothes is not present.

Change House with Swimming Pools (By A. H. Sawyer).—A new change house has recently been completed at the Raimund mines of the Republic Iron & Steel Co., near Bessemer, Ala. It has some novel features which should make its description interesting. The building, illustrated in Fig. 407, is constructed of concrete and brick, 121 ft. 9 in. long by 37 ft. 3 in. wide and contains 365 lockers, 280 of which are for colored employees and 85 for white employees, including 5 in the foremen's office. The dimensions of the building give 11.5 sq. ft. per man. If the portion occupied by the offices and swimming pools be excepted, the space per man is 7.3 sq. ft., which is less than in most change houses.

The swimming pools, one for white and one for colored employees, are 27 ft. long and 17 ft. wide with a maximum depth of 8 ft., and hold 21,292

gal. up to the overflow. The water is heated by live steam admitted through pipes into the bottoms of the pools. At one end of the pool and at the water line there is imbedded in the concrete a 3-in. pipe with a slot running its entire length. This serves as an overflow pipe and also as a means to cleanse the surface of the water. No one will be allowed to use the swimming pools unless he has the permission of the mine physician and has previously taken a shower bath.

The lockers are 15 × 15 × 60 in., with perforated bottoms, arranged in rows of 20 each; they rest on wrought-iron supports which also carry seats 12 in. wide. Under each double row of lockers, extending its full

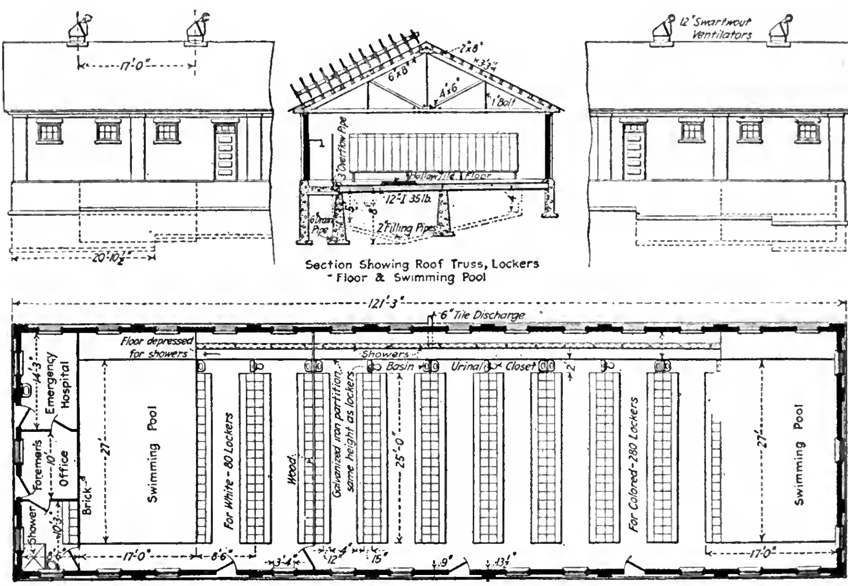


FIG. 407.—SOUTHERN CHANGE HOUSE WITH SEPARATE ACCOMMODATIONS FOR WHITES AND NEGROES.

length, is a radiator made of eight 1¼-in. pipes, giving a total of 1300 sq. ft. of radiating surface for heating the building. Live steam under 3-lb. pressure is used. Each pair of rows of lockers has wash basins, closet and urinal at the rear, separated from the shower-bath compartment by galvanized-iron partitions. Two sanitary drinking fountains are provided, one in the white and one in the colored section.

There are 19 showers; the water enters a mixing chamber through hot- and cold-water pipes, fitted with lock valves, below which are stop and waste cocks, so that the quantity of water for each shower can be regulated by the attendant, preventing the use of an excessive quantity of water. Above the mixing chamber is a valve for turning the water on and

off. In the main part of the change house, there is one shower for every 20 men. It is estimated that 8 gal. of hot water will be sufficient for each bath. This water is heated by live steam, in a closed heater situated in the basement of the building. Room has been allowed here for the installation of an independent boiler if it should at any time seem desirable. The building is lighted by incandescent lamps, 12 in the large room and a drop light in each of the small rooms. An emergency hospital 14 ft. 3 in. by 8 ft. is situated just off the foremen's office and is fitted with a wash basin, a first-aid outfit and an ambulance which can be handled easily by one man.

United Verde's New Change House.—The old change house at the United Verde mine at Jerome, Ariz., was replaced by a modern steel

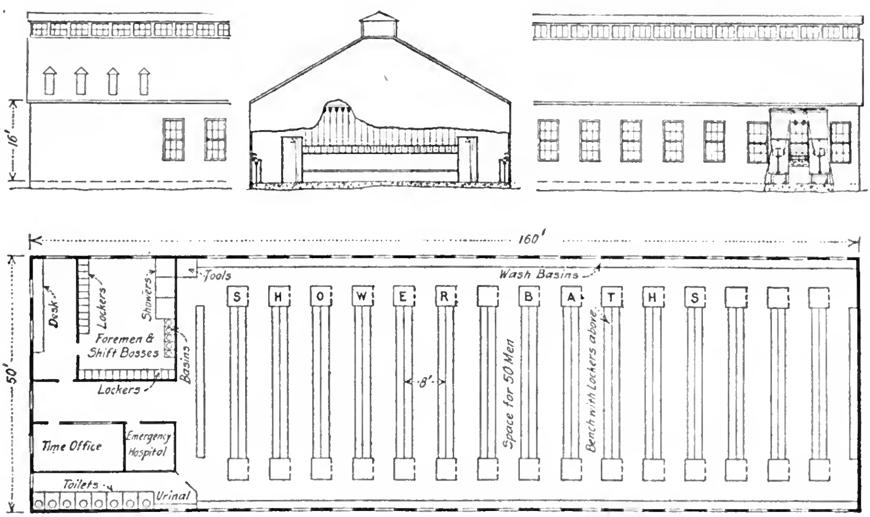


FIG. 408.—CHANGE HOUSE OF THE UNITED VERDE, JEROME, ARIZONA.

change house embodying the latest ideas in sanitation and privacy, such as chain "lockers" of the Continental type and individual shower baths. This type of locker was adopted in the new change houses built by the Copper Queen at Bisbee and by the Butte & Superior at Butte; the new United Verde change house is based on the experience of the above two companies, but certain changes were made to suit the local conditions and to increase individual privacy.

The building is 50 × 160 ft. with accommodations for 800 miners and 25 foremen and shift bosses. It has a steel frame with corrugated-iron sides and roof; the reinforced-concrete floor is slightly elevated and supported on the steel frame so that the building may be leveled in case of subsidence of the ground. The site is at the mouth of the 50-ft. adit,

which on completion of the change house became the main entrance to the mine for the miners. The mine time office is situated just inside the entrance to the change house, and adjoining is an emergency hospital equipped with stretchers and other supplies for first-aid work. The foremen's office and change room are opposite and the remainder of the building is devoted to accommodations for the miners.

The main room has a row of 15 double benches and two single benches at each end. The locker system consists of the chain lockers already mentioned, supplemented by small metal lockers, 15 × 14⅜ × 12 in., for shoes, soap, etc. These small metal lockers are fastened to the back of the benches within easy reach, and the chain suspending the clothes is locked behind the hasp of the metal locker by a single padlock. The miner's clothes are suspended from a cast-iron frame having six hooks. Between each of the double benches there is space for 50 men.

At the ends of the benches there are shower baths with galvanized-iron sides and a canvas front curtain. Along the sides of the building there are 144 enameled-iron basins equipped with the usual rubber stoppers, and spring faucets to prevent waste of water, which is important here. The basins are arranged in batteries of nine and empty into a trough that serves an entire battery and discharges into a trapped drain pipe. The basins are hinged and may be swung back to permit the cleaning of the trough by an attendant after the miners have left. The concrete floor of this room is 1½ in. higher at the center and has gutters along each side, so that the floor may be flushed and readily cleaned. The building is provided with steam heat, and is well ventilated through the monitor at the top of the building. There is an attendant on each shift; these attendants are trained in first-aid work and also look after the emergency hospital.

Homestake Two-story Change House (By B. C. Yates).—The Homestake Mining Co., of Lead, S. D., recently completed and put into service a change house for underground men, which presents some new features in construction and equipment that may be of interest. The building is situated near the Ellison hoist, and was designed to accommodate about 600 men. The walls are of hollow concrete blocks in two courses, giving a total thickness of 16½ in. The ground floor is of plain concrete laid on a broken-stone base, and the second floor is of reinforced concrete, supported on I-beams, the beams covered on the under side to protect them from rust, as well as from any fire that might occur in the clothing. The roof is of steel covered with Carey's patented cement roofing, laid on 1½-in. sheathing. There is no ceiling under the roof trusses. Three 30-in. Globe ventilators along the ridge of the roof assist in carrying off foul air and lessen the smell which is always present where many damp working clothes are kept. The window-glass area is about 17 per cent.

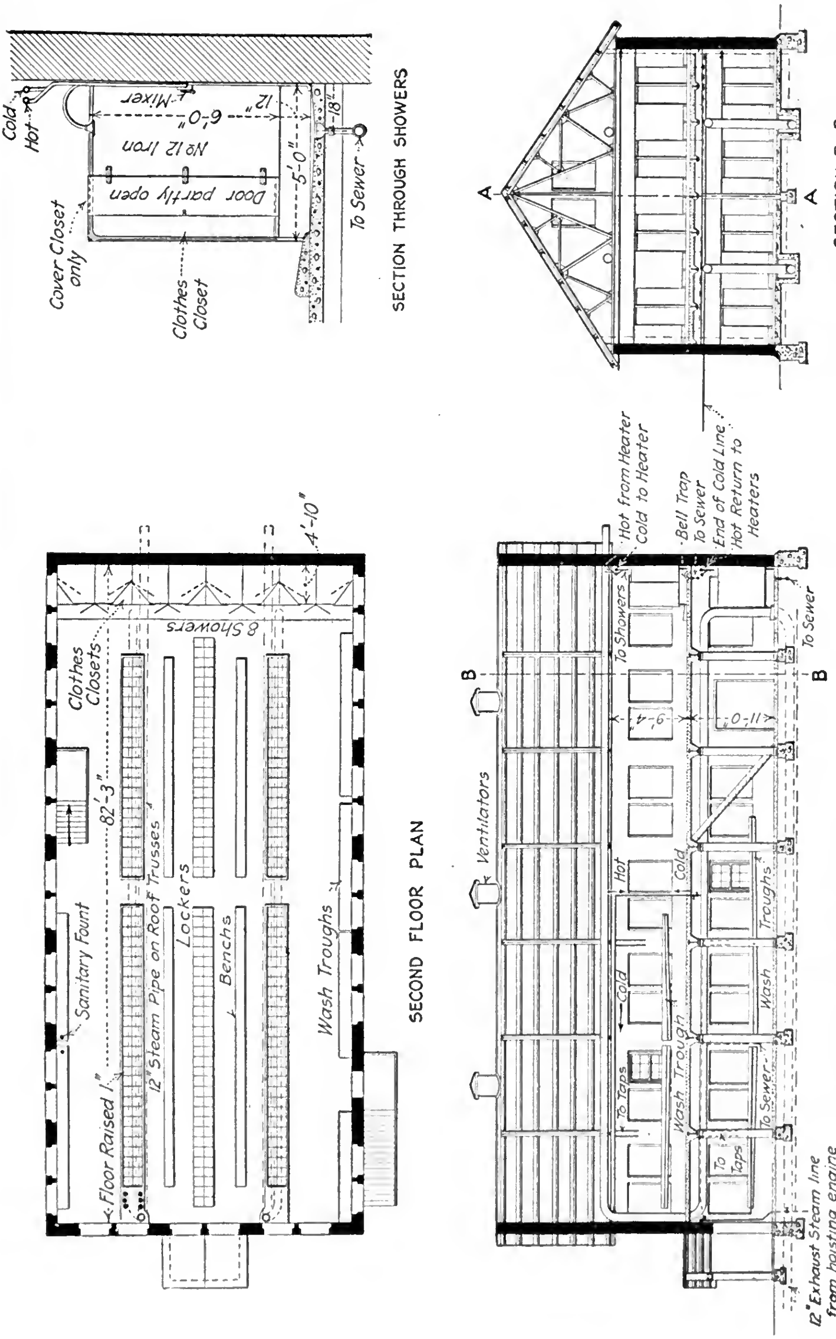


FIG. 409.—SECTIONS AND SECOND-FLOOR PLAN OF THE NEW HOMESTAKE CHANGE HOUSE.

of the entire wall area and is distributed in such a way as to give efficient lighting except in the end in which the showers are located. Here the building wall forms part of a retaining wall, built against the natural rock bank at the end of the excavation.

The arrangement of the building is shown in Figs. 409 and 410. It is heated by exhaust steam from the hoisting engine, the steam being carried to the rear end of the building in two 12-in. pipes laid under the ground floor in conduits covered with steel plates. Here the pipes are turned and carried up to the under side of the second-floor beams, returned to the front end of the building, passed up through the second floor to the bottom chord of the roof trusses and finally returned to the rear end and passed out through the wall. Water for wash basins and

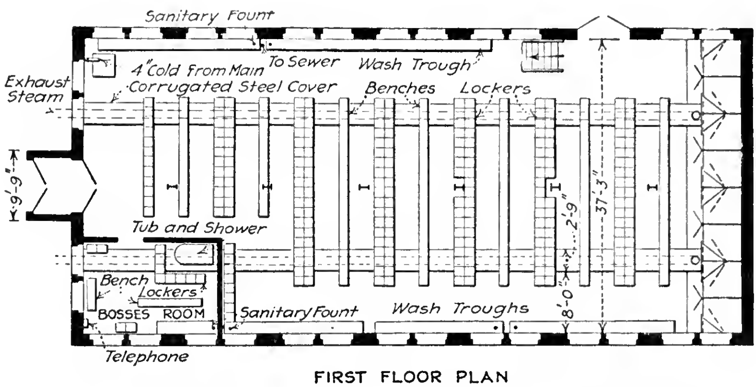


FIG. 410.—FIRST-FLOOR PLAN OF THE HOMESTAKE CHANGE HOUSE.

showers is heated by steam in a large tank heater in the hoist building nearby. There are 16 shower stalls with partitions made of galvanized iron supported on a framework of angle iron. A patented mixing valve called the Niedecken Mixer, made by the Hoffman & Billings Mfg. Co., Milwaukee, Wis., is used to control the supply of hot and cold water to the showers.

The lockers are all steel, 15 × 15 × 72 in., with perforated doors, a top shelf for hats and small articles, and hooks under this shelf for clothes, the bottom serving as a shelf for boots. Each man is allotted one locker and furnishes his own lock and key. The galvanized-iron wash-troughs are set along the walls under the windows and hot- and cold-water faucets are spaced about 24 in. along the trough. Each man is supplied with a wash basin, which he keeps in his own locker. A room in one corner of the first floor is used by the shift bosses. In this room are lockers, a bath-tub and shower, a wash trough, a desk and a telephone. This room offers opportunity for the bosses to talk over their work privately

and permits their being reached by telephone before going down into the mine.

LATRINES, ETC.

Sanitary Underground Latrine (By W. B. Hambly and A. E. Hall).—At most mines the problem of the underground privy is serious. If the men are allowed to go to the surface excessive time is lost; the portable privies frequently used are generally unsatisfactory, if the waste from the privies is allowed to run into the sump, unless pumping is frequent, there is danger of having the material collecting on the sump bottom. Fig. 411 shows the design of a privy which keeps the waste materials entirely separate from the sump and is a unit in itself.

The device is simple. It consists of a piece of 12-in. cast-iron pipe or any sort of pipe with a cap on each end, the caps bored and threaded for a 4-in. pipe. The intake is at the bottom of one end and the discharge at the top of the other. A small auxiliary pipe, inserted at one end, to be used as a run-off after flushing, leads to the sump. Two saddle flanges are placed as shown, the 12-in. pipe being cut to match. The saddle flanges should be 10-in. A 10-in. nipple about 4 in. long is placed in the saddle and on the nipple is screwed half of a flange union. A wooden seat is attached to the top by the bolt holes in the union. Flap valves must be placed inside the main pipe at the saddle flanges to prevent overflowing during the process of flushing. A valve is placed in the water column leading from the pump and connections are made on either side of this valve; these connections are also provided with valves as shown. The valves are all placed so that they come inside the pump house; only the pump man has access to them and he is responsible for the flushing. When the privy does not need flushing, valve No. 1 is open and No. 2 and No. 3 are closed. The discharge from the pump is then up the shaft direct. Valve No. 3 is a gate valve, but the other two can be globe valves. When it is desired to flush the closet, valve No. 1 is closed and No. 2 and No. 3 are opened. The water then takes the indirect course through the closet, carrying the waste to the surface. When it has been flushed, valves No. 2 and No. 3 are closed and valves No. 1 and No. 4 are opened. As shown, there are only two seats, but any convenient number can be put on.

Septic Tank for Underground Latrine (By H. G. Pickard).—The septic tank shown in the accompanying sketch was designed for the Mond Nickel Co.'s Frood Extension mine at Sudbury. The Ontario mining law requires the provision of underground closets, and the Provincial health laws forbid the discharging of untreated sewage into running streams. At the best the ordinary box closet is a nuisance, and there are many objections to running untreated sewage through the pumps to the

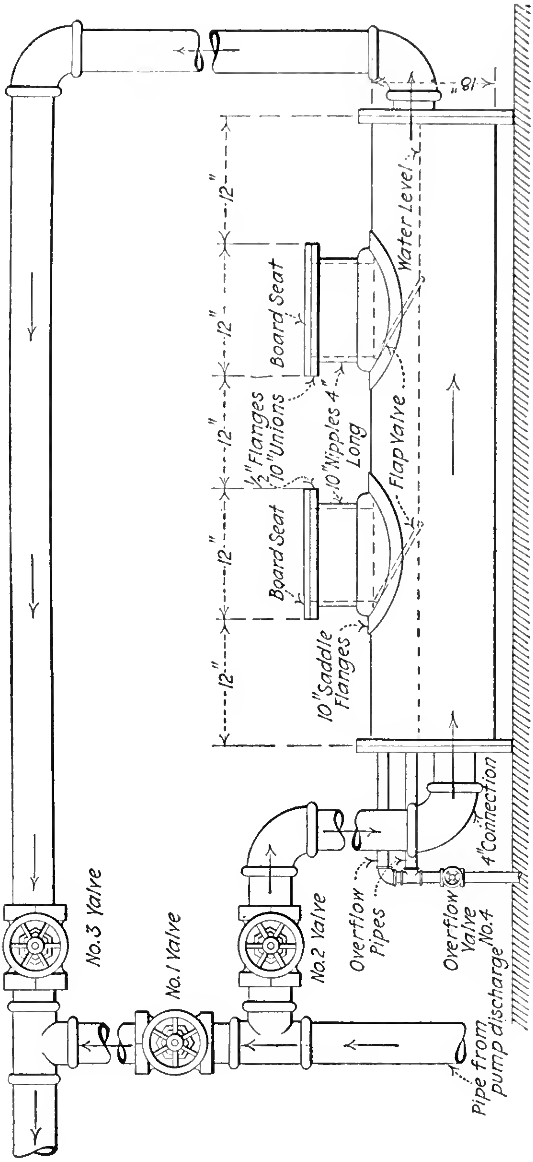


FIG. 411.—LATRINE BUILT OF PIPE AND FLUSHED FROM PUMP COLUMN.

surface. To overcome the objections to these types of closets, it was decided to introduce a septic tank. The stools flush automatically, and discharge through a pipe into the tank, which may be located at any point provided there is sufficient fall from the tank to the pump sump. Fig. 412 shows the ideal location, since the excavations for sump and tank may be taken out at one operation.

With septic tanks, as ordinarily constructed, it is found necessary to clean out from once or twice a year the sediment that is deposited. To obviate the necessity of doing this cleaning by hand, iron pipes having their ends drawn to flat nozzles are inserted in the concrete lining of the tank. The pipes are bent so as to deflect a stream of water toward the centers of the chambers. A bypass from the pump connects with the

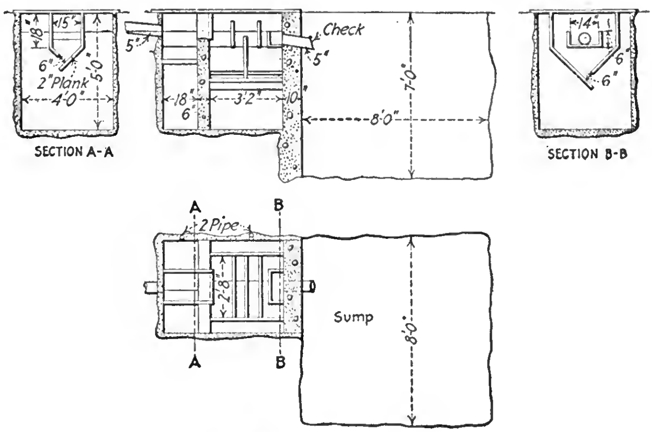


FIG. 412.—ARRANGEMENT OF UNDERGROUND SEPTIC TANK.

wash pipes, and sediment may be washed into the sump, where it will remain long enough in suspension to be discharged by the pump before settling. The tank shown was designed to accommodate a force of 150 men per day.

Concrete Latrine at the New United Verde Smelting Plant.—At Clarkdale, Ariz., a special effort was made to maintain sanitary conditions both during the period of construction and later. At the new smelting plant of the United Verde Copper Co. a concrete latrine was provided, connecting with the general sewerage system which discharges into a septic tank. The building is approximately 8 × 20 ft., sheathed with No. 22 corrugated iron. The general arrangement is shown in the accompanying drawings by Repath & McGregor, Fig. 413, who designed the new works. The bottom of the concrete trough has a slope ¼ in. to the foot and the flow of water is regulated by the valve at the head end, while at the outlet there is a double trap consisting of a 4-in.

lead pipe and an 8-in. tile trap. At the top of the trough on each side is a 1-in. perforated pipe that constantly sprays the sides. The rail is a polished and rounded 2 × 6-in. plank fastened to the concrete by ½-in. hook bolts. To insure cleanliness there is a back rail placed 1 ft. 8 in. above the seat.

Septic Tanks at Clarkdale, Arizona.—To avoid pollution of the Verde River, the United Verde Copper Co. constructed a septic tank to take the sewage from the new smelting plant and from the "construction town" at Clarkdale. Another septic tank of similar size was constructed for the main residence portion of Clarkdale, the present construction camp becoming the "Mexican town."

The septic tanks are 53 ft. long by 26 ft. 6 in. wide, constructed of reinforced concrete. Each tank required 450 cu. yd. of excavation and 115 cu. yd. of concrete, the mixture for the latter being 1 :2½ :5. The

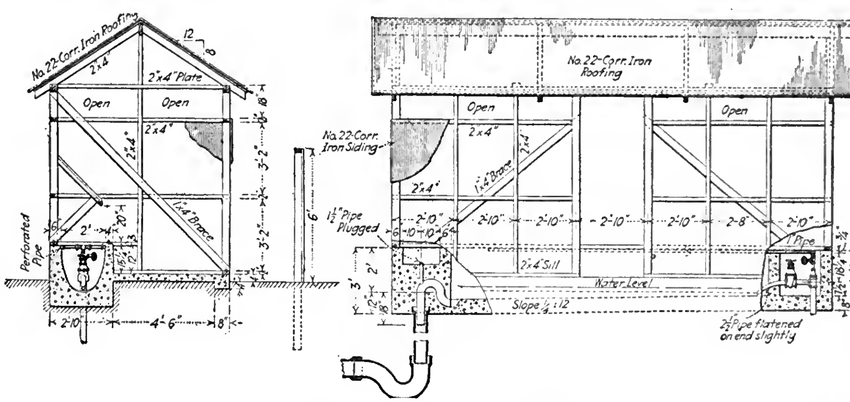


FIG. 413.—CONCRETE LATRINE, UNITED VERDE WORKS, CLARKDALE, ARIZONA.

approximate cost of such a tank is from $1500 to $2000, depending upon the locality in which it is constructed. The tank, designed by Repath & McGregor for the Clarkdale works and construction town, is shown in Fig. 414. The tank is 7 ft. deep and is designed in two units so as to permit occasional inspection or cleaning. Following the receiving compartments, there are two series of four working compartments each. The first compartment is 16 ft. long, the second 12 ft., the third 10 ft., and the fourth 8 ft. The connection between the various working compartments is through twelve 3-in. pipes in the partition walls about 3 ft. from the bottom of the tank and at an angle of 45° for the purpose of directing the flow to the top of the next compartment. The tank walls and partitions are 10 in. and the bottom 6 in. thick; the tank is covered with reinforced-concrete slabs, 6 ft. 4 in. long, 15¾ in. wide, and 3 in. thick. When for any purpose entrance to the tank is desired these con-

crete slabs may be removed, but during the normal operation the slabs are sealed by a covering of earth 12 in. thick at the center line of the tank and sloping to 8 in. at the sides. No air is permitted to enter the septic tank, but it is necessary to provide for the removal of gases formed within the tank, and this is accomplished by a 2-in. ventilating pipe in the partition walls of each compartment, the gases being removed through a 3-in. ventilator near the top of the last compartment.

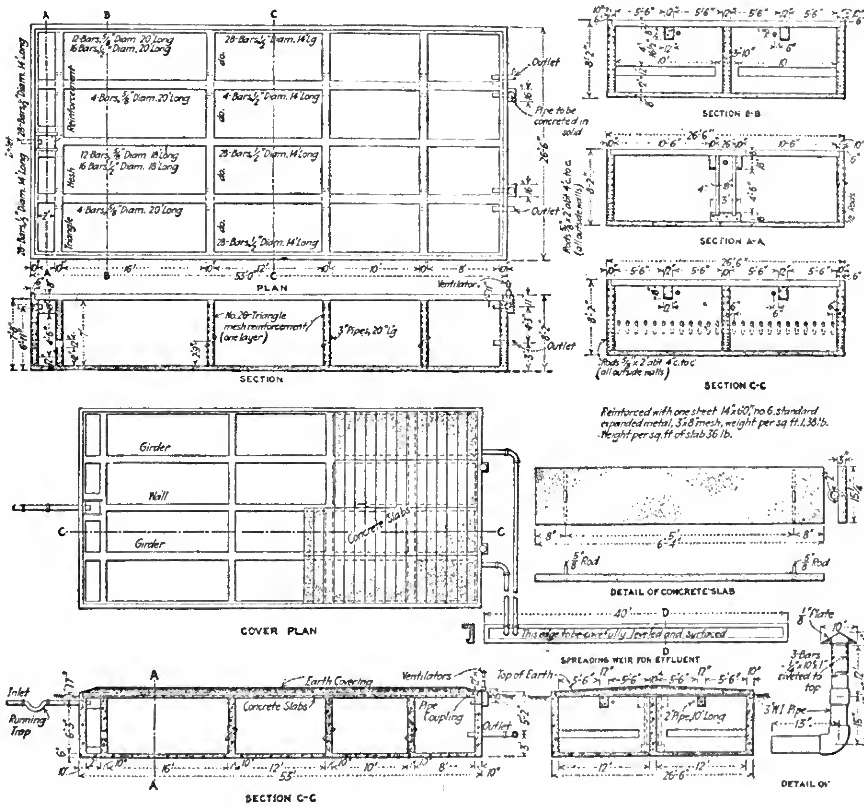


FIG. 414.—SEPTIC TANKS FOR SEWAGE DISPOSAL, CLARKDALE, ARIZ.

The sewage enters the septic tank about 6 in. below the top through an 8-in. trapped iron pipe that delivers the material to a receiving box, a trough on each side connecting with receiving compartments, 2 ft. by 11 ft. 4 in. by 7 ft., from which the sewage passes through 12-in. slots about 2 ft. above the bottom to the two series of working compartments. The disintegration of the organic material in the septic tank is brought about by the anaërobic bacteria, *i.e.*, those which live and act only in the absence of air. The material entering the tank separates into three

portions: A solid portion which sinks to the bottom, a liquid layer in the center, and a portion which floats at the top of the liquid. By the action of the anaërobic bacteria the solid floating and settled portions are gradually converted into liquid and pass through the pipe connections in the partition walls to the succeeding compartments and out through a 4-in. pipe about 2½ ft. above the bottom of the last compartment. Most of the work is accomplished in the first compartment and the succeeding compartments are gradually reduced in length. The outlet pipe from each series of compartments leads to a spreading weir, 40 ft. long by about 2 ft. wide, over which the effluent passes in a thin layer to a gravel bed about 40 ft. square where purification is finally accomplished by aërobic bacteria and sunlight. While the organic material in the sewage is rendered innocuous by this treatment, no attempt is made to deodorize the released gases. The septic tank should consequently be built at some distance from any residences or other inhabited buildings. There are many mining communities where sanitary conditions could be vastly improved at comparatively small expense by the installation of a system of this sort.

FIRST AID

First-aid Bandage Roller (By E. W. R. Butcher).—Fig. 415 shows a bandage roller used by the Republic Iron & Steel Co., on the Mesabi range, in its first-aid work. The different classes are required to do prac-

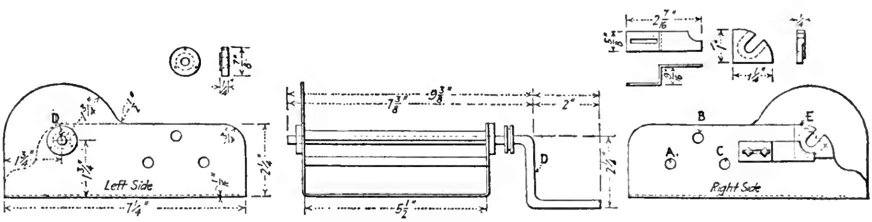


FIG. 415.—DEVICE FOR ROLLING BANDAGES.

tice work in their course of training and the ordinary method of rolling bandages by hand was found too tedious for the large number which had to be rolled. This device was designed from a wooden one somewhat similar to it which was being used by one of the range hospitals. The bandage to be rolled is brought under the rod *A*, over the rod *B*, under the rod *C*, and given about three turns around the rod *D*. The left hand is then placed on the bandage, pressing it against the table to which the roller is attached, and it is rolled to any degree of tightness by regulating the pressure of the hand on the bandage. One edge is kept close to the far side of the frame. The bandage can be removed by taking out the pin *E*,

which allows the crank and rod to come out, so that the bandage can slide off the rod.

Resuscitation.—The U. S. Bureau of Mines in *Technical Paper* 77, gives the report of the committee on resuscitation from mine gases. The committee was composed of doctors. After reviewing the committee report, the Bureau engineers made the following recommendations in cases of gassing or shock:

In case of gassing, remove victim at once from gaseous atmosphere. Carry him quickly to fresh air and immediately give manual artificial respiration. Do not stop to loosen clothing. Every moment of delay

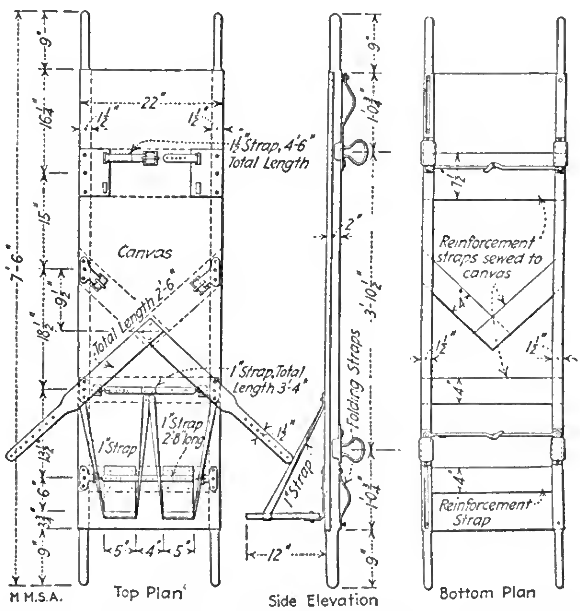


FIG. 416.—THREE VIEWS OF NEW JERSEY ZINC CO. STRETCHER.

is serious. In case of electric shock, break electric current instantly. Free the patient from the current with a single quick motion, using any dry nonconductor, such as clothing, rope, or board, to move patient or wire. Beware of using any metal or moist material. Meantime have every effort made to shut off current. Attend instantly to the victim's breathing. If the victim is not breathing, he should be given manual artificial respiration at once. If the patient is breathing slowly and regularly, do not give artificial respiration but let nature restore breathing unaided.

In gas cases, give oxygen. If the patient has been gassed, give him pure oxygen, with manual artificial respiration. The oxygen may be given through a breathing bag from a cylinder having a reducing valve,

with connecting tubes and face mask, and with an inspiratory and expiratory valve, of which the latter communicates directly with the atmosphere. No mechanical artificial resuscitating device should be used except possibly one operated by hand that has no suction effect on the lungs. Use the Schaefer or prone pressure method of artificial respiration. Begin at once. A moment's delay is serious. Continue the artificial respiration. If necessary, continue two hours or longer without interruption until natural breathing is restored. If natural breathing stops after being restored, use artificial respiration again.

Do not give the patient any liquid by mouth until he is fully conscious. Give him fresh air, but keep his body warm. Send for the nearest doctor as soon as the accident is discovered.

Underground Stretcher (*Bull.*, Mining & Metallurgical Society of America).—Fig. 416 shows a stretcher for underground use employed at the Franklin Furnace mine of the New Jersey Zinc Co.

MISCELLANEOUS

Fire-fighting Pipe Lines at Mount Morgan (By B. Magnus).—To deal successfully with a fire discovered in its initial stage, it is essential that a supply of water under a good pressure should be immediately available at the scene of the outbreak. To make this possible, fire hoses and suitable connections to the network of pipe lines which supplies water for use with the rock drills have been provided for the Mount Morgan Mine in Queensland, Australia, at convenient points on the various levels. Every working face has water pipe laid to it, which by means of a ½-in. rubber hose, supplies the necessary spray for boring and allaying the dust, consequently this pipe-line system is always full of water. The pipes, however, are only 1½ in. in diameter, much too small to supply sufficient water if a fire once gained a firm hold. To furnish immediately abundant water, the underground air mains are connected with the surface reservoirs, so that within a few minutes after an alarm is given the 8-in. air mains can be conveying water to the place of the fire. This is effected by shutting off the air and opening the water-connection valves, thus giving the water free access to the air mains.

The accompanying section, Fig. 417, will assist in coming to an understanding of the system. To reduce the excessive head of water which would result if no break were made in the circuit, open tanks are placed at several points, underground and on the surface, at *B*, *D* and *E*, as indicated on the section. These tanks are provided with ball float valves which automatically keep them full. The reservoir marked *A*, situated on the original hillside at the south end of the opencut, supplies water to the tank *B*. For fire-fighting between the 450-ft. level and the 574-ft.

level, the pressure is obtained from *B*, which is situated 87 ft. above the 450-ft. level; on the 574-ft. level, therefore, the head is 211 ft. A second reservoir marked *C* stores the water for the lower levels. It supplies a tank marked *D*, provided with a float valve, which, in its turn, supplies the tank *E* on the 650-ft. level. From this last, the water is drawn for the 750-ft., the 850-ft. and the 950-ft. levels.

One great advantage of using the air pipes is, that a supply of water is assured at every working place. In the event of fire, the rock-drill hoses, which are 1 in. in diameter, can be used as water hoses. To supplement these hoses there are on the main levels, connections to the standpipes for 2-in. canvas hoses. Printed directions regarding the opening and closing of valves are posted in conspicuous places, so that the uninitiated may be able to manipulate them. Once the water is turned into the air mains, the valves regulating the supply of air in its ordinary course will serve to direct the water to wherever it is needed.

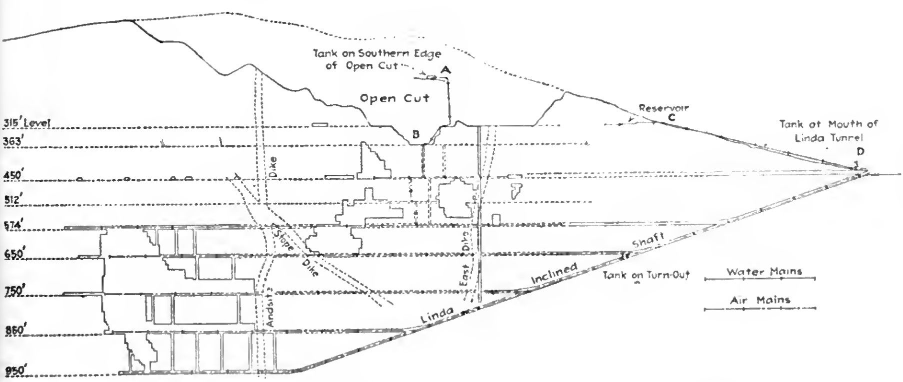


FIG. 417.—SECTION SHOWING PIPE LINES FOR FIRE FIGHTING.

While such an arrangement as this would be dangerous if there were a possibility that men might be cut off by a fire in such a way as to be left dependent on the compressed air to escape suffocation, the layout of this particular mine renders such an occurrence exceedingly unlikely. The two main entrance and exit shafts are at the extreme southeast end of the workings, which trend in general northwest and southeast. At the far northwest end of the workings, a ventilating shaft supplies an emergency exit. It is considered that these openings, in connection with the interlevel winzes and ladderways, reduce to a minimum the possibility of trapping and suffocation.

Sanitary Fountain Made from Cask (By E. C. Carter).—Fig. 418 shows the water casks used in the mines of the Gold Hill & Iowa Mines Co., at Quartzburg, Idaho. Small wine casks are used for this purpose, and furnish a sanitary and inexpensive method of supplying water to the

different parts of the mine. The carmen generally send these kegs to the surface with the first car out on each shift, thus keeping the water fresh and cool. By giving the keg a slight slant toward the front sufficient force is obtained to drive the water from the ¼-in. pipe until the keg is practically empty.

[If it were found that the arrangement as presented offered a temptation to suck out the water when it was low, a head could be obtained at all times by dropping the opening below the bottom of the barrel, using three ells instead of one. If this were done, merely opening the stopcock would always insure an immediate flow of water.—Editor.]

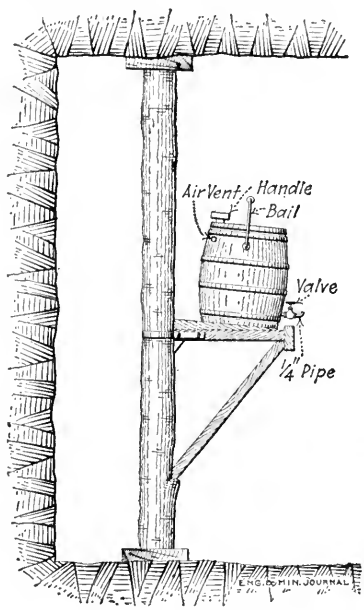


FIG. 418.—Sanitary underground drinking cask.

Home-made Shower Bath (By E. W. R. Butcher).—A novel shower bath recently constructed by the volunteer fire department of the Schley mine, at Gilbert, Minn., contains interesting features of value to anyone desiring to construct a shower bath at small cost. The hall near which the men carried on their training was some distance from the mine, and it was not possible to use mine steam for heating the shower-bath water; consequently some other scheme had to be devised, and it was decided to use electric current.

A small building, 8 × 14 ft., situated in back of the fire hall, had been used as a woodshed; it was cleaned out and made into the bathhouse. A hot-water tank was erected, consisting of a 6-ft. piece of old 15-in. water-column pipe. This pipe was closed at both ends with a ⅝-in. piece of

plate, the upper end being braced with two 2½-in. angle irons. The arrangement of the piping is self-explanatory in Fig. 419. An electric heating coil surrounds a 1-in. pipe. This pipe is covered with a single layer of uncut mica, on which is wound the heating element, consisting of 25 ft. of No. 22 Calido Element wire. The vertical pipe is attached to two horizontal nipples with Navy ground-joint-union connections to provide easy access to the tank; the nipple connects to the hot-water tank. A mixing chamber for hot and cold water is provided, consisting of a piece of 5-in. pipe, the ends of which are closed by two plates welded on. The sprinkler of the shower itself was made from a ¹⁄₁₆-in. copper plate. The upper part was cupped out and the lower part punched thoroughly with small holes.

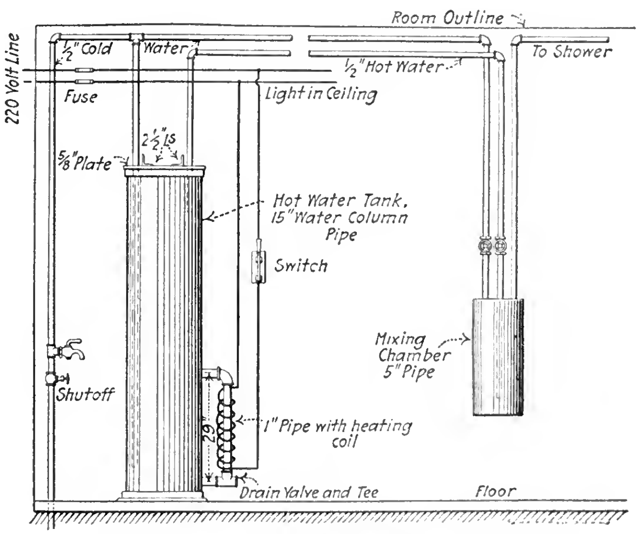


FIG. 419.—LAYOUT OF HOMEMADE SHOWER BATH.

Two rubbing appliances were made, a one-hand and a two-hand. The two-hand was made from ¼-in. flexible shafting and the one-hand from ¼-in. solid shafting. The shafting in both cases was strung its whole length outside the handles with soft-rubber washers drilled eccentric which gave an excellent kneading effect. Towels were made from cement sacks which were ripped and washed thoroughly. The men stated that they were just as good as bath towels.

Removable Chute-spray.—A recent act of the Nevada legislature made it compulsory to put sprinkling devices on all underground ore chutes where dry and dusty ore is handled. In Fig. 420 is shown a convenient form of spray in use at the Mason Valley mine for laying the dust when loading cars. All stoping is done by the shrinkage method and

as large boulders are unavoidably covered in the stopes, frequent blasting at the chute mouths is necessary; the spray was designed to meet this condition. Water is piped into the mine through a 1-in. main, and a 1-in. pipe is brought up beside each chute to a height of 4 ft. To this pipe, by suitable fittings, as shown, is connected a ½-in. globe valve; a ½-in.

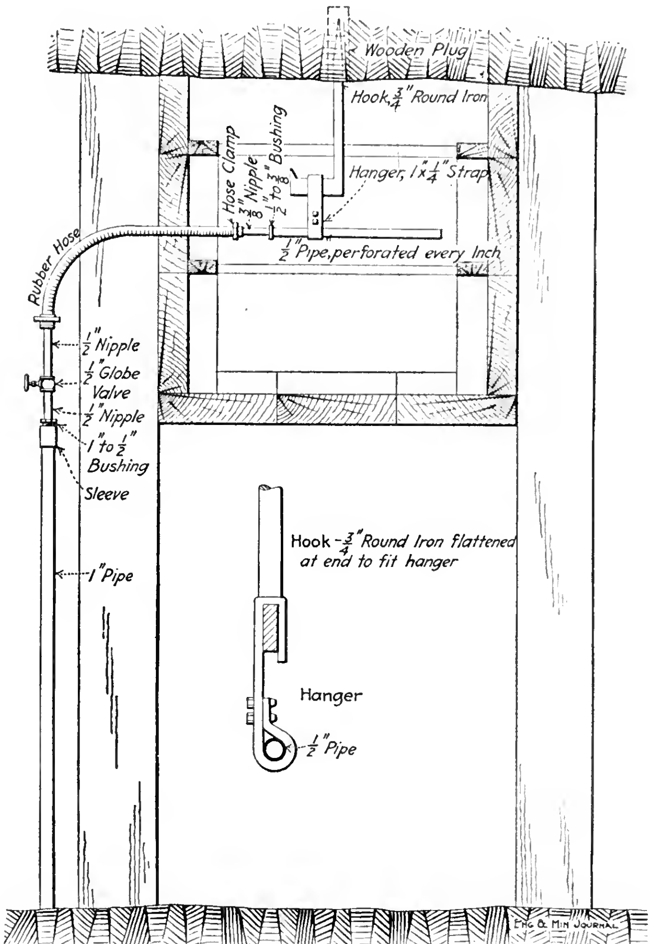


FIG. 420.—ARRANGEMENT OF SPRAY AT CHUTE MOUTH.

rubber hose; and then the spray proper. The spray consists of a 2-ft. length of ½-in. pipe, plugged at one end and perforated by small holes, placed in a row and at 1-in. spaces. To the spray pipe is bolted the hanger. This is a piece of 1 × ¼-in. iron, bent to fit the hook. The hook is made of a short length of a ¾-in. round iron, one end bent at right

angles and flattened, the other end pointed and driven into a wood plug in a drill hole in the roof of the drift. The flat hook and the fit of the hanger prevent the spray from swinging, so that the water is always thrown in the right direction. This spraying device is simply and cheaply constructed; the water can be easily regulated by the carman and is always directed to the right spot; the spray can be swung to one side when it is necessary to blast the chute, and the hook, the only part liable to injury by blasting, can be easily replaced.

XIII

DRAINAGE AND VENTILATION

Pumps and Parts—Air Lifts—Sealing off Water—Ventilating Devices

PUMPS AND PARTS

Unwatering Shaft with Horizontal Turbine Pumps (By L. C. Moore). —The Dexter mine, 7 miles west of Ishpeming, Mich., was shut down in 1896 and the pumps were pulled. In the spring of 1914 the Cleveland-Cliffs company, now in control, decided to unwater it for exploration purposes. The shaft is 12 ft. long and 3½ ft. in minimum width and dips at an angle of 54°. It is approximately 600 ft. deep on the incline and has eight levels. Most of the water enters on the second or 200-ft

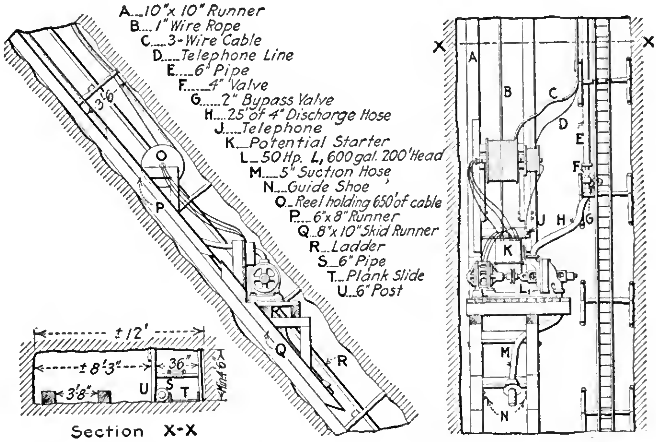


FIG. 421.—LAYOUT OF PUMP IN SHAFT, ARRANGED FOR LOWERING.

level, but streams were encountered also on the third and seventh levels. The pumping equipment when the job was completed consisted of one 200-ft. head, 600-gal. per minute turbine pump and two 400-ft. head, 300-gal. turbine pumps, with motors, starting boxes, etc., complete. The motors were 50-hp., 2200-volt, 60-cycle machines, interchangeable to make the outfit more flexible. The shaft pump was mounted on a skid, as shown in Fig. 421, while the lead-covered cable reel was mounted on a lighter skid just above the pump. The smaller reel shown contains the duplex telephone wire, one telephone being placed on the skid and one

in the engine house. All communication and signaling were carried on with the telephone.

Pumping operations started May 3 and the mine was completely unwatered on July 17. When the 200-ft. level was reached the 600-gal. pump was set off, a 300-gal. pump installed in its place and the work in the shaft completed with the latter. The third pump was installed on the third level to take care of the shaft water when the shaft pump got beyond its head and could not throw to the surface. The incoming water amounts to approximately 400 gal. per minute. Storms made some trouble; one breakdown of the transmission line caused an eight-hour delay and a drowned pump. This happened on a Thursday, but by the next Monday the pump was recovered and the motor dried out and back in commission again. The advantages of the system lay in the ease with which the pumps handled the water with a variation in head from 50 ft. to 200 ft.; the low head room required; the light foundations needed for satisfactory operation; the quick changes that could be made in position; and the reliability of the turbine pumps under all conditions.

Economical Pump Arrangement (By Howard S. Lee).—At the Rainbow mine in eastern Oregon it became necessary to provide an additional pump to take care of extra water encountered in development work on the lowest level. In order to be able to handle surges of water and still not incur the expense of a pump with a capacity much greater than the normal flow, an installation, as shown in Fig. 422, was devised by E. A. Hamilton, master mechanic. While the arrangement is not entirely original, it may be of interest to those having similar problems on account of its low initial cost and its operating economy as regards both labor and power.

The pump already in use was an Alberger, four-stage centrifugal, with a capacity of 100 gal. per minute, direct connected to a motor with a speed of 3600 r.p.m. This pump was connected to storage tanks on the hillside about 80 ft. vertically above the collar of the shaft, and the water thus pumped was used in the mill or allowed to overflow to waste. The new pump is a Goulds triplex with a capacity of 175 gal. per minute. It is connected by an endless balata belt to a 25-hp. motor. In order to place the belt and motor back against the wall and out of the way, the crankshaft and countershaft were turned end for end. An idler was applied to the top or slack side of the belt to give greater pulley contact and eliminate vibration due to belt whip. Both the pump and motor were placed on wooden foundations to save the cost of concrete. The sills were well braced and securely bolted and there is no appreciable vibration. The motor foundation was built 21 in. higher than the pump foundation, which will permit 2½ ft. of water on the floor of the station before the motor is submerged. The pump was set above the sump to eliminate

unnecessary bends in the suction and make it as short as possible. A by-pass to the sump and a check valve were placed in the discharge column of the triplex pump and the flow of water governed by an angle float-valve in the bypass. Between the two sumps there is a ditch 2 ft. deep and the float works within this range. By this arrangement the centrifugal pump can never be without water and only the excess passes through the triplex pump. It would have been possible to connect both pumps to the same water column, but as there was already another column in the shaft which discharged into an old adit about 160 ft. vertically below the storage tanks, this was used. Now only such water as is needed in the mill is pumped to the higher level; the excess is discharged at the lower level, thereby saving power.

To regulate further the flow of water, a bulkhead was placed in the main crosscut which may be closed when the power is off or when it becomes necessary to reduce the amount of water entering the sump.

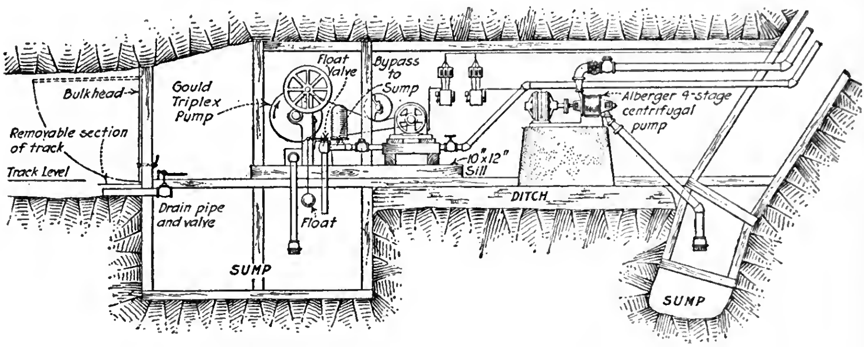


FIG. 422.—LAYOUT OF PUMPS, SUMPS, DITCH AND CONTROL VALVE.

This bulkhead was made of two thicknesses of 1-in. lumber and provided with a felt gasket next to the timber. It is hinged at the top and when closed may be fastened with bolts and wing nuts. A 6-in. pipe and valve were provided so that all water can be drained before the bulkhead is opened. A short section of track may be removed when the bulkhead is closed. When in place the track is fastened by four loose pins. By this arrangement the flow of water can be regulated absolutely. In case of power interruption it is not necessary to use the auxiliary steam pump to keep the electrical pumps from being flooded and the expense of fuel, firemen and pumpmen in these emergencies is eliminated. In addition to this the triplex sump acts as a settling sump and no grit can enter the centrifugal pump. With the float-valve arrangement the water is maintained at a constant level and no pumpman is required.

Homemade Hand Pump.—A convenient hand pump for incidental work around a mine can be made at slight expense. Fig. 423 represents

such a pump, using a $2\frac{1}{2}$-in. pipe. The pump in its simplest form will not suck, but the plunger must work in the water. By the attachment of a check valve at the bottom of the pipe, however, a suction lift could be had. For its construction, three $\frac{3}{8}$-in. rods are welded to the end of a $\frac{1}{2}$-in. rod, the latter of the same length as the pipe through which the pumping is to be done. The three rods are turned or swaged to $\frac{3}{16}$ in. at the lower ends to pass through the plunger and are threaded. The plunger pieces are held against the $\frac{3}{32}$-in. collars on the rods by nuts. The plunger consists of two brass disks $\frac{1}{4}$ in. thick with a piece of leather between. The lower disk fits loose in the pipe. The upper disk is slightly smaller. Thus the leather will have a slight upward turn on its edges and will be tight in the pipe when the plunger rises and loose enough when it descends, to permit easy working. A $\frac{3}{4}$-in. hole through the plunger is covered with a leather flap riveted to the plunger on one side and weighted down with a brass cap. This forms the working valve.

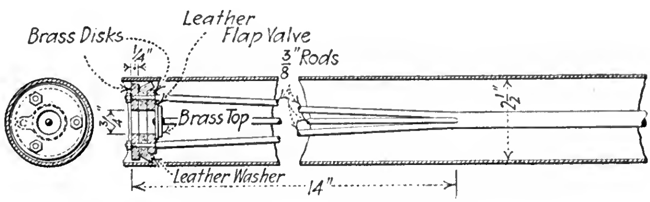


FIG. 423.—SIMPLE HAND PUMP FOR $2\frac{1}{2}$-IN. PIPE.

Hand-controlled Compressed-air Pumping Barrel (By Arthur O. Christensen).—The homemade arrangement here described has been found useful in bailing out winzes or underhand stopes where the amount of water is not sufficient to pay for installing a pump, while the lift is too high to bail by hand; and also in situations where a pump is not desirable because of blasting, or lack of room, or cost of moving and installing, or because none is to be had. The barrel, rigged as illustrated in Fig. 424, can readily be put in place and operated with the same air and even the same hose as is used to run the drills. Into the upper side of the barrel is screwed a 1-in. nipple which has a 1-in. to $1\frac{1}{4}$-in. bushing for a hose spud. Into the bottom is screwed a 2-in. nipple. A few weights, such as pieces of track rails, drill steel or tripod weights, are bound to the bottom to act as ballast and sink the barrel, thereby hastening its filling.

In operation, the air-discharge cock is opened, allowing the barrel to fill and sink. As soon as air ceases to issue from this cock, it is closed and the compressed air is turned on. The water in the barrel is now forced up the discharge hose until air begins to issue with the water, showing the barrel to be empty. When so much of the water has been pumped out that the barrel no longer fills, the hose is unscrewed from the top, a funnel

inserted and the last of the water bailed by hand into the barrel. The hose is then screwed on again and the apparatus operated as before. The check valve between the barrel and the discharge is not necessary unless there is a possibility of the discharge line siphoning back the discharged water while the barrel is filling again. This operation might be made automatic, but a regular trap or ordinary pump would be more serviceable in such a case. The device, lifting water 30 ft. with an air pressure of 100 lb. per square inch, and an average difference of 2 ft. in water

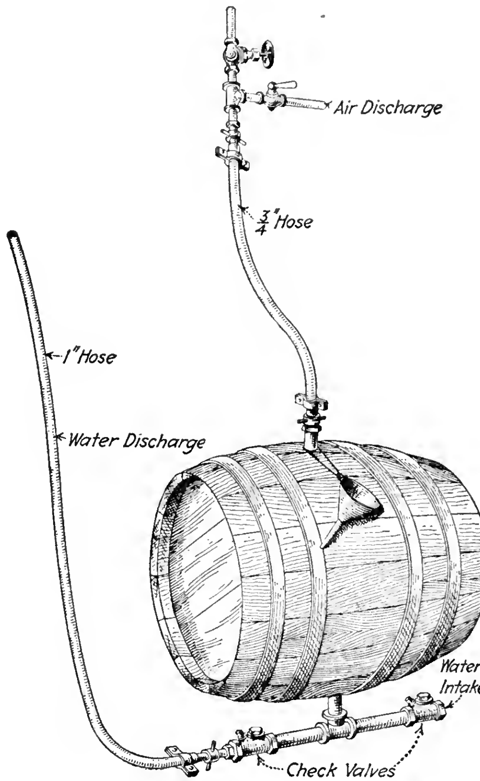


FIG. 424.—VALVES AND CONNECTIONS FOR PUMPING BARREL.

levels between that inside and that outside the barrel, should handle 25 to 30 gal. per minute. With a 2-in. discharge pipe or a 10-ft. submergence, this rate would be nearly doubled.

Cast-steel Pump Valve.—A valve designed and patented by J. P. Matthews, of the Rogers-Brown Ore Co., is illustrated in Fig. 425. It was found in the operations of this company on the Cuyuna range that the solid rubber valves on the underground pumps were expensive in the first place and required constant changing. The new valve is designed

to consist principally of steel; a casting is made as light as possible by being built in skeleton form and reinforced with ribs and two circular grooves are left to receive packing rings of rubber or other available

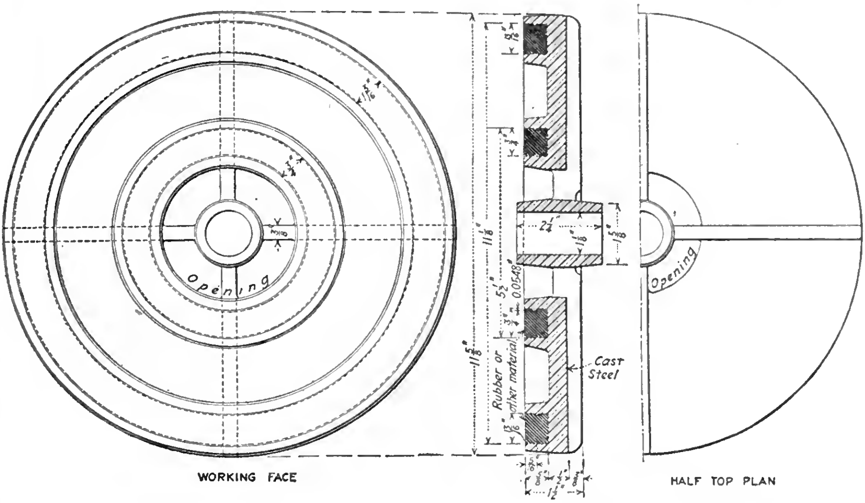


FIG. 425.—CAST-STEEL SKELETON VALVE TO HOLD RUBBER PACKING.

material. The packing soon wears flush with the steel and thereafter wear on the whole valve is much slower. The arrangement obviously saves rubber in the first place and also gives the valve a longer life. It is stated to be eminently satisfactory in service. The valve illustrated is

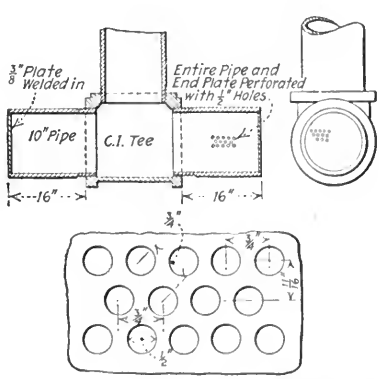


FIG. 426.—SUCTION INLET FOR UNDERGROUND PUMP.

for the smaller type of Prescott pump used by the company; for the larger pump, however, the only change is in the dimensions.

Suction for Station Pump (By H. L. Botsford).—Fig. 426 shows a suction for a 24-40-7 × 36-in. pumping engine. The area of the ½-in.

holes is five times the area of the pipe. The capacity of the pump is 1200 gal. per minute or 600 gal. for each side. This requires a velocity of 150 ft. per minute in the suction pipe, and of 30 ft. per minute through the ½-in. holes. It is good practice in the design of pump suction to limit the velocity to 200 ft. per minute and to make it less than that if the pipe is longer than 25 ft., or has many elbows.

AIR LIFTS

Combination Pump and Air Lift (*Power*).—An extremely ingenious method of connecting an air-actuated mine pump is illustrated in Fig. 427. The device was installed to eliminate freezing troubles in the exhaust passages. It was not practicable to reheat the air; so the connections shown in the illustrations were made. The pump was a 12 and 6 × 13-in. machine, working against a 400-ft. head under an air pressure

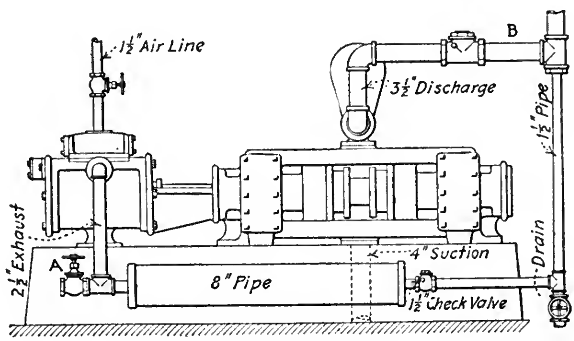


FIG. 427.—PUMP EXHAUSTING INTO DISCHARGE COLUMN.

of 90 lb. The exhaust was piped to a receiver made of an 8-in. by 4-ft. pipe. A connection *A* was provided for exhausting into the open if desired. The receiver was connected to the discharge *B* of the pump, and in this connection, a check valve was set to prevent the return of the water to the air-end of the pump. The air working in the pump cylinder nonexpansively left the cylinder at approximately 90 lb. It was thus able to force its way into the pump column provided this was not filled to a point to produce a head greater than 90 lb. Once mixed with the water, the air acted like an air lift, forming a mixture in the pump column much lighter than water, expanding in its upward course and assisting in raising the water. With the proportions of the air end, the water end and the lift here given, there could be no further possibility of the air not being able to exhaust against the pressure of the discharge column.

The freezing difficulty was effectually overcome, but the idea would seem to be capable of more important development. A mine pump

with cylinder ratios designed for a certain lift could have its possible lift thus increased, something often desirable where sinking pumps must be adopted for station work. Furthermore, a permanent installation could be designed with the ratio of the cylinder diameters so adjusted to the lift and quantity of water to be handled, that the air should be exhausted at the discharge end of the pump column at atmospheric pressure and be made to do all the work possible, thus utilizing all the potential energy in the compressed air.

Air Lifts for Shaft Unwatering (By Arthur O. Christensen).—A shaft can usually be unwatered more quickly and cheaply with an air lift than with pumps. It is, therefore, advisable, before allowing a mine to flood, to connect the air line to the discharge of the pump line as near the bottom

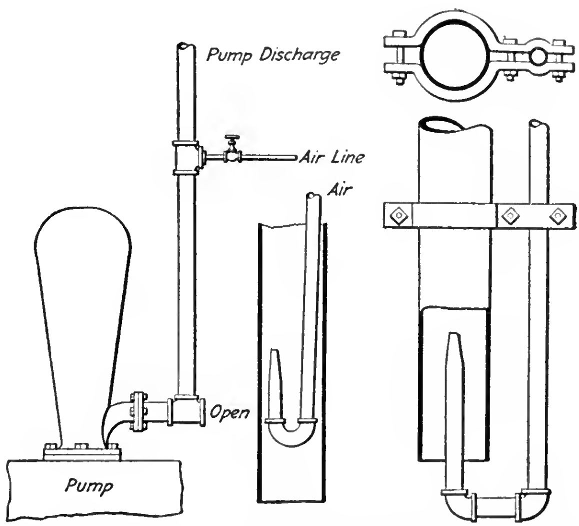


FIG. 428.—CONNECTIONS FOR AIR LIFTS UNDER VARYING CONDITIONS.

of the shaft as possible, preferably a few feet above the pump, as indicated in Fig. 428. If there is a tee where the line joins the pump this should be left open. If there is not a tee, the line should be disconnected from the pump. To start unwatering, compressed air is simply turned into the air line and the pump-discharge line begins to work as an air lift. The diameter of the air line where connected to the water pipe should be about one-fourth the latter. Even if the pump is one which can be started while drowned, so that an air-lift attachment seems unnecessary, still it is desirable to have the connection made as described, for there is no knowing how long the pump may be submerged or whether it will work when the air supply is turned on. In such a case, however, it is not necessary to make any opening in the discharge line, since, if the air lift is

to operate without the pump, it can suck water through the pump valves. Both lift and pump can be used, one helping the other. The air lift relieves the pump of a portion of its work by diminishing the head in the discharge, or, looking at it from the opposite standpoint, the pump aids the lift by supplying its suction end with water at a pressure which may be greater and cannot be less than would be the case were the pump not to act. Conversely if it is found that a pump is not able to lift its water to the point of discharge, the suction end being all right, a small amount of air admitted to the discharge line just above the pump will cause it at once to speed up.

In most cases, however, it is too much to expect a retiring management to leave a mine in such shape that it is necessary only to start up the compressor in order to unwater the property. The question is how to unwater the shaft as it stands. Frequently there is a pipe line running straight down the shaft. Whether the bottom of this be open or closed, and whether there be a pump attached or not, the air lift can be operated, so long as the line is straight enough to allow a pipe to be run down the inside. Fig. 428 represents the lower end of the inner pipe. If no return elbow is at hand, two elbows connected by a close nipple will answer. The nozzle is made by heating one end of a short pipe and forging it down to about three-fourths or one-half its original diameter. This arrangement can suck water through a drowned pump, although, of course, not so readily as if there were no such obstruction. Where a suitable pipe is in the shaft but is closed by a valve or otherwise, a charge of dynamite can be lowered by a small pipe within the larger, and the charge exploded by electricity—one ordinary dry cell will set off an exploder—thus breaking the pipe. The saving by using an air lift for the job is assumed to be more than the cost of repairs to the pipe line later on.

In case no suitable pipe for the water discharge is found in the shaft, a simple air lift can be dropped down a vertical shaft or can be lowered in an inclined shaft either on the truck or simply on a plank laid across the rails. A 2-in. and a 1-in. line make a convenient size for such a lift, as is shown in Fig. 428. At intervals of from 15 to 30 ft. the two lines are strapped together as indicated. This arrangement will operate either vertically or on an incline. As soon as it is noticed that the ratio of water to air being discharged has decreased considerably, the lift must be submerged more, or, if this is not possible, the height of its discharge must be diminished by installing a pump as near the water as possible to throw the water from the lift to the surface. With 100 lb. of air pressure, such a lift as indicated can raise water 200 ft. with a 20-ft. submergence, or can lift 50 ft. with almost no submergence at all. The air issuing from the nozzle serves as an ejector to suck up and carry the water along with it. Air lifts can be compounded to raise water any heights with any

submergence, down to almost nothing, but in general a pump is preferable to such a complicated arrangement.

Six-inch Pipe for Air-lift Sump (By M. J. McGill).—While sinking at the Silver King Consolidated, trouble was had from a small flow of water, about 15 gal. per minute, from the 1100-ft. station and about one set below. Instead of installing a pump, it was decided to rig an air lift. The water had to be pumped to a winze about 1800 ft. in from the shaft,

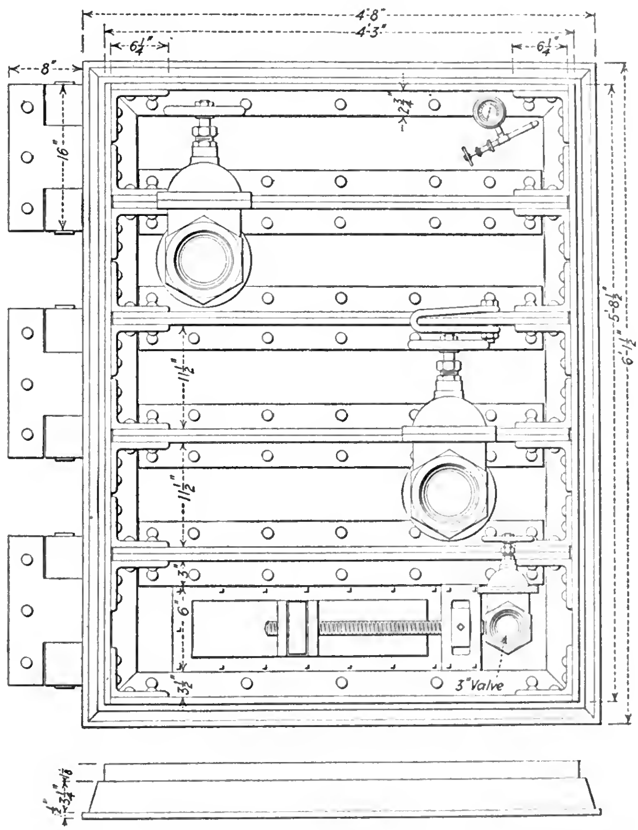


FIG. 429.—CONSTRUCTION OF DOOR AND ARRANGEMENT OF VALVES.

the winze collar being about 20.5 ft. above the station level. In the shaft 34 ft. of 6-in. pipe was hung from a point one set below the station and blanked at the lower end except for a 1-in. drain with a gate valve. Within 4 in. of the end of some 2-in. pipe, several ¾-in. holes were drilled, and just above this a ¼-in. nipple with an ell on each end was tapped in. In the outside ell a ¼-in. pipe was screwed. The ¼-in. and the 2-in. pipes, bound together, were lowered into the 6-in. pipe. The ¼-in. was connected to the compressed-air line, compressor pressure

being 100 lb.; the 2-in. pipe was connected to an old 2-in. air line which led to the winze. At about 100 ft. in a tee was put in on the 2-in. line and a valve to relieve the air pockets. The flow of water was directed into the 6-in. pipe. With the air-inlet valve open about one-sixth of a turn, the water kept about 14 in. below the top of the 6-in. pipe, except when the miners turned on the air to blow smoke from the shaft bottom.

SEALING OFF WATER

Built-up Iron Water Door (By R. R. Heap).—The draining of an ore run near Miami, Okla., involved the use of a bulkhead door to control the flow of water to the pump. This door is illustrated in Fig. 429. The drainage head was 50 ft., and the door was built to withstand a pressure in excess of requirements, which gave it extreme solidity of construction. Its width and height were determined by the opening necessary for tramming the ore to the shaft, at least 5 ft. 6 in. in height and 4 ft. in width. The Webb City and Carterville Foundry & Machine Works of Webb City, Mo., made the plans and built the door. It was considered safest to regulate the volume of water flowing to the pump by means of a series of valves in the door, which were more than sufficient in total area to pass the entire volume of water; it would not have been advisable to regulate the volume of water by opening the door to any required width. The drawings show the construction and installation so clearly as to require pointing out only its most important features.

The door and frame were installed in shale. They were both made of heavy boiler iron with all joints securely riveted in place. The contact joints were beveled and lined with a metallic packing. To insure that the setting of the frame should be watertight, the drift at the point of installation was enlarged by cutting a space 4 ft. deep and 4 ft. wide on both sides of the drift in the floor and in the back as shown in Fig. 430. A set of timbers was then put up, to which the frame was bolted level, and the open 4-ft. spaces were filled in with a rich mixture of concrete, flush with the top of the sill, with the insides of the posts and with the bottom of the cap. The further precaution was taken of extending the concrete filling, flush with the insides of the timbers, for two sets in front and behind the door frame. The trackway through the door was made in a removable section, the 3 × 6-in. ties being fitted into daps cut out of the sills, so as to make a close joint and one easily and quickly removable.

Two 6-in. and one 3-in. iron body, brass mounted, wedge-pattern gate valves were screwed into flanges riveted to the inside of the door, using nipples just long enough to give room for the valve wheels. At the bottom a slide door was installed, arranged to be opened and closed by means of a heavy air-drill square-threaded feed screw and nut, the nut tightened

on the screw by a ⅝-in. setscrew and held in place in a slot 2¼ in. wide. The drawing shows the slide door tightly closed. Its maximum opening gives a space 6 × 12 in., more than sufficient with the 3-in. valve to pass the entire flow of water.

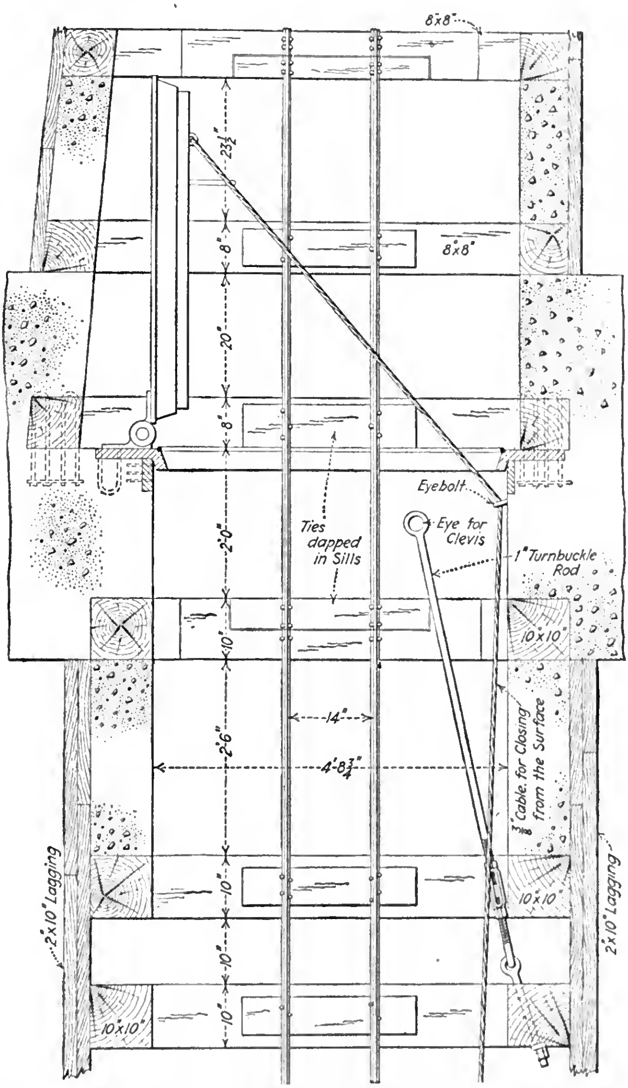


FIG. 430.—DOOR IN DRIFT, CONCRETE LINING AND REMOVABLE TRACK.

The door was held tightly closed with a 1-in. turnbuckle rod, fastened with an eye-bolt on one end to a post of the third set back and at the other end to the door with a heavy clevis of 1⅛-in. round iron. This clevis was

fastened to the door at one of the upper joints by a ¾-in. king bolt slipped through the eyes in the clevis and the ⅞-in. hole in the door joint. With the clevis in place a few turns of the turnbuckle to the right securely tightened the door; a few turns to the left gave ample play easily to pull out the clevis bolt; after slipping the clevis from the large eye in the end of the rod, the latter would swing down out of the way on the side of the drift and the door would open. An offset in the drift allowed the door to swing back out of the way. A means of closing the door from the surface was also provided; a small ⅜-in. cable was fastened to the door at the top and passed through two eye-bolts, one on the side of the drift and one at the corner of the shaft. On the surface it was fastened to a post of the

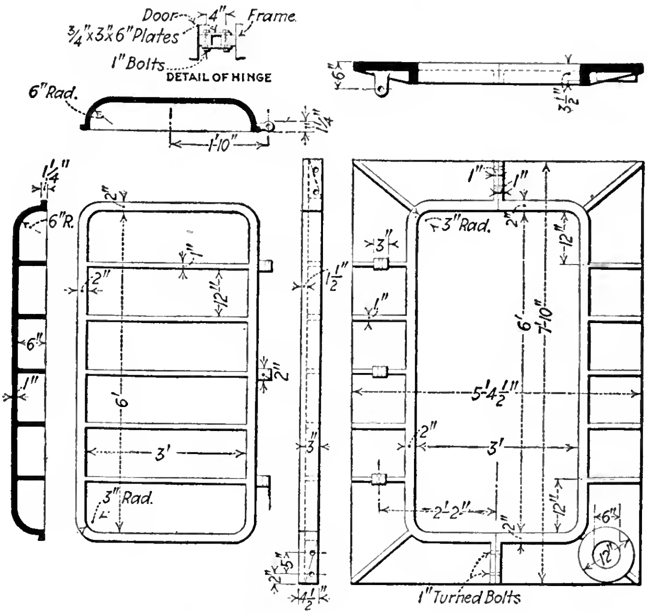


FIG. 431.—DOOR AND FRAME TO CONTROL UNDERGROUND WATER.

headframe. By means of a loop in this cable it could be hooked into the hoisting cable snap-hook and tightened with the hoist. The value of such a door, as permitting the successful use of a single pumping unit, is evident. The series of valves is of great importance as an efficient means for regulating the flow of water and of eliminating the danger of flooding the pump, which might occur under certain conditions were the size of the opening of the door itself the only gage.

Cast-iron Door for Mine Water (By H. Beard).—Cast-iron doors to resist the pressure of water underground are particularly useful in controlling the flow where a sudden rush of water is expected or pumping is interrupted by trouble with the pump or with the power. The door and

frame, shown on the left and right respectively of Fig. 431, were designed for use by the Alvarado Consolidated Mines Co., in Parral. One in use under a head of 75 ft. of water is quite tight and others were to be installed to resist heads of 125 to 140 ft. For these greater heads it is advisable to reinforce the frame with stay-bolts to the roof, floor and walls of the drift. It is estimated that the door will resist a pressure of 100 lb. per square inch. The door was cast at the local foundry in one piece, the frame in two pieces. The dimensions are as shown and the estimated weight is 2640 lb. for all the pieces. The pieces of the frame are planed where the flanges meet and are bolted together. In one of the lower corners a round flange is cast on with a 6-in. opening for water discharge. The door is self-sealing, there being no provision for holding it against the frame other than the pressure of the water. For this purpose, the faces of both frame and door are planed, and a ½-in. groove ¼-in. deep is planed in the face of the door to take ½-in. square canvas packing. To enable the door to come to a proper seating, the hinges are made with links of ¾-in. plate, as shown, giving some play. To set the frame in the drift, a small recess in the rock is cut on all four sides and the frame held in this with concrete. In cutting this recess, great care should be taken in blasting and all loose pieces removed in order to prevent leakage. Sufficient room should also be cut out in the back to allow the concrete to be easily tamped into place on the top of the frame.

Concrete Bulkhead Under 200-lb. Head (*Bull.*, American Institute of Mining Engineers).—At the Hibernia magnetite mine in New Jersey, it was found desirable to separate the old workings from the new and to allow the former to fill with water to the 850-ft. level. For the 16th level, a truncated-wedge dam was adopted. The pressure side of the dam is of greater area than the back, so that the resultant action is similar to driving the wedge. By cutting generous skewbacks in the walls, roof and floor, this becomes in reality an invisible arch. The wedge feature tends to compress the materials in the bulkhead, thereby adding to its imperviousness. Concrete was chosen as the material. To lessen the labor and simplify form construction, straight forms were placed on both the front and back of the dam, making the arch invisible. As ordinary concrete is not impervious under a head of 200 lb., it was decided to waterproof the concrete by facing the entire pressure side with a 3-in. layer of "Impervite," a waterproofing compound. This facing was carried up with the concrete to insure a perfect bond. A manway through the dam was provided to permit inspection or repairs.

The relative positions of an old bulkhead and the new bulkhead are shown in Fig. 432. The old bulkhead leaked to the extent of about 16 gal. per minute. The five pipes through it permitted pumping to the surface the water from behind it. The new bulkhead was designed to continue

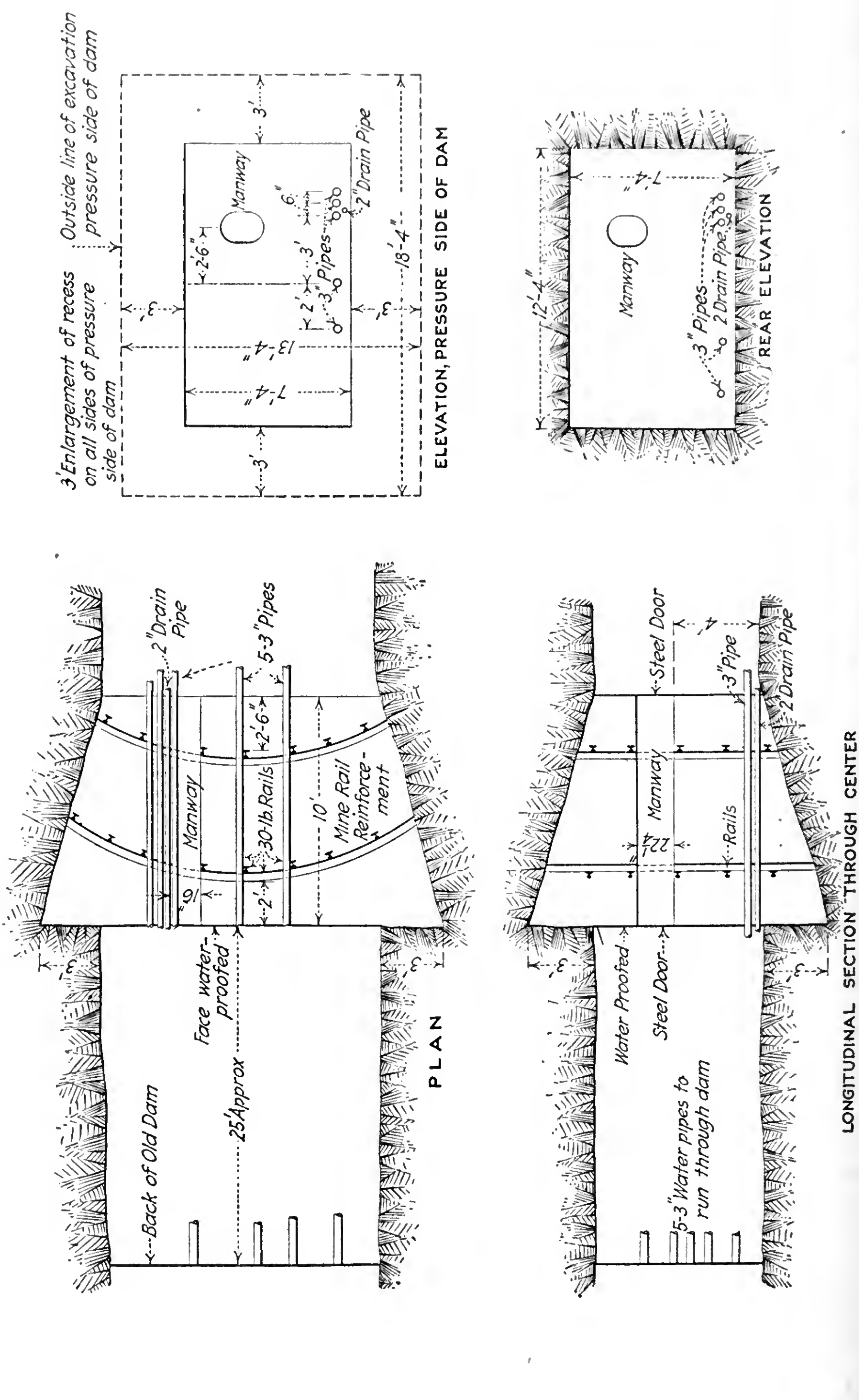


FIG. 432.—PLAN AND LONGITUDINAL SECTION AND END ELEVATIONS OF DAM.

this function of drainage if it were necessary, and hence prolongations of the pipes were carried through it. The horizontal rails used for reinforcement were curved to the radius of the invisible arch. Great care was taken thoroughly to coat all metal surfaces with mortar. A manway pipe was provided through the bulkhead. It was decided to use a 1:2:4 mixture. The cement was Atlas portland; a local sand, carrying less than 3 per cent. of foreign matter, was obtained; the broken stone was a gneiss, the result of former milling operations.

In drilling the recesses for the bulkhead, care was taken to point the holes so that the excavation would coincide with the design in form and dimensions. At a distance of 25 ft. from the old dam, holes 36 to 39 in. in length and spaced 1 ft. apart were drilled in the sides, roof and floor, at right angles to the course of the drift, using stoping and hand drills. Thirty-five feet from the old bulkhead, a series of holes 4 ft. in length and from 1 to 1½ ft. apart were placed slanting to conform approximately with the inclinations of the skewbacks. These holes were burdened with only about 1 ft. of ground. Under ordinary conditions, longer holes would have been drilled, but the proximity of operating pumps made extraordinary precautions necessary for their protection during the shooting. A group of six holes was shot at a time. A third series of holes was drilled slanting to conform with the deeper portions of the recesses. When blasted, this series broke evenlya t the line of the 3-ft. hole first drilled, and the resultant recess conformed almost exactly to the figure determined upon, while the total excavation agreed with the original estimate of 60 yd.

The materials required were stored close to the shaft collar. Due to the lack of space on the level, the matter of delivering the materials without interrupting work was troublesome. Sand and stone were sacked on the surface. The empty bags produced as the cement was used augmented the 200 old cement bags purchased for the sacking. A small night crew lowered much of the material required for the next day's work. The 10 curved rails were bent on the surface over a 14-ft. radius. The bulkhead forms were built of 2-in. undressed lumber, with 6- to 10-in. round posts for studding and braces. The forms were thoroughly braced and were wired to stiffen them. The interior faces of the forms were covered with tar paper, and the junction of the forms with the rock was plastered with a 1:1 cement mortar on all sides. The pressure-side forms were carried to the roof of the level at once, but did not extend into the recess. The recess was thoroughly cleaned of loose rock and washed down, and all the reinforcing material, the pipes and the manway were placed in position before the concreting was started. The floor and sides of the recess were plastered with a 1:1 cement mortar before placing the concrete.

A batch of concrete contained ⅔ cu. yd. The sand was first placed on the mixing platform and the heaps flattened down. On this was emptied the cement, and these two materials were thoroughly mixed and flattened out before receiving the stone. This mixing took place about 12 ft. from the front form of the bulkhead. Enough water was used to make a wet mixture. Two men did the first mixing and turned the mass, then passed it on to the next two, who again turned it, passing the finished concrete to the last two men at the mixing board. These men shoveled directly into the form. In this manner, while each two men received a short rest of a few minutes between batches, fresh material was being placed on the starting end of the mixing platform while the men nearest the form were still disposing of the concrete mixture. This also insured a thorough mixing. One man remained in the form to level off each batch. The best day's work consisted of placing 12 yd. of concrete. The waterproofing compound was carried up as a 3-in. facing, level with the concrete. An even thickness of the waterproof layer was maintained by the use of three forms of a 3/64-in. plate, 6 ft. long by 6 in. wide, fitted at the upper corner with 3-in. spreading bolts. These forms, placed across the entire width of the face, were raised 3 to 4 in. at a time, and enough concrete was then shoveled against them to keep them in place. The almost semiliquid waterproofing compound was mixed on the level and was carried to the forms in buckets. Before leaving at night, sharp stones of about 100 lb. weight were set at least 6 in. apart in the concrete mass. This made a strong bond, and before concreting the next day, this rough surface was freshly plastered with a thin 1:1 mortar. As the roof was reached, false forms were placed, and the work was finally finished in tightly bonded dovetailed blocks. Throughout the work, the leakage from the old dam passed through the 2-in. drain pipe of the bulkhead.

Seven 2-in. grout pipes, four on the pressure side and three on the opposite side, were placed in the concrete as the work neared completion. They were all placed near the roof and directed to the places most difficult to fill with concrete. As the work had to be hurried, but a day and a half elapsed after completion of the cement work before grouting was begun. The grout consisted of one and one-half parts of sand to one part of cement made fluid with water-dissolved "Impervite." A mine-made grout gun was used and the grout was forced successively into the several pipes by means of air under the pressure of 85 lb. per square inch. When the grout was forced through the different pipes, its ejection through the other pipes indicated that the greater voids were filled. As the gun connections were changed, those pipes giving the greatest discharge were plugged, and the discharge was finally limited to one pipe. This, too, was filled and plugged. The first day's grouting was allowed to set over night, and the following day all the pipes were again tested. This time

there was no communication between the pipes, and as little or no grout could be forced into any one of the pipes, the grouting was considered most satisfactory. For three weeks the new bulkhead did not receive any load. During this time the 2-in. drain pipe was left open. The bulkhead was tested by pumping water up to the pressure of 160 lb. into the space between the old and the new bulkheads through the 2-in. drain pipe. The results were entirely satisfactory; the total seepage amounted to only ½ gal. per minute at first, and this small leakage subsequently stopped almost completely.

A cheap class of labor was employed exclusively, the men receiving $2 per 10-hr. shift. Following is a table showing the cost of the work. The interference caused by the necessity of keeping two large pumps in operation within 50 ft. of the bulkhead was perhaps the greatest cause for the high cost. The labor cost of lowering materials was also high.

Summary of Costs

	Total	Per cu. yd.
Labor	$790.00	$13.17
Superintendence	130.00	2.17
Transportation	50.46	0.84
Materials	503.88	8.38
Totals	$1474.34	$24.56

Plugging Water Channels into Mine (By J. E. Reno).—The Little Mary mine is situated on a flood plain of the North Fork of Spring River, 3 miles northwest of Neck City, Jasper County, Mo. The mill was erected and started to operate in the summer of 1910; mining was carried on successfully until some time in March, 1911, when the river, through sink holes in its bottom and a solution channel following the contact of the shale and limestone as indicated on the map, Fig. 433, broke into the mine workings. The water came into the workings from all sides and its volume was so great that the mine was flooded and work had to be abandoned until the high water in the river subsided, when various schemes for stopping up the sink holes were tried. A drought of about four months' duration followed the wet period and while the river was extremely low coffer-dams of burlap sacks filled with soil from the river bank were built around the sink holes. The holes were then filled with hay and dirt and covered with boulders. During this time, June 1 to 17, the mine was dewatered. The channel had still not been closed, since muddy water came into the mine, and it was evident that dirt was not the material with which to choke off the water. It was next attempted to seal the sink holes with rock and concrete, but in August when another

flood came, new sink holes developed and the water flow into the mine for about two months was as heavy as ever. In the meantime more pumps were installed, capable of handling the water, which had increased from 700 to about 3000 gal. per minute. For two weeks more the former methods were tried at a further cost of $1000, but without success.

The plan was then devised of running a tailing flume from the mill to the river, a distance of 600 ft., and extending it to the various sink holes. This cost $600 but did away with carting the tailings. The tailings contained a good deal of cementing material and were found to be a cheap and efficient material both for the coffer-dams and for filling the

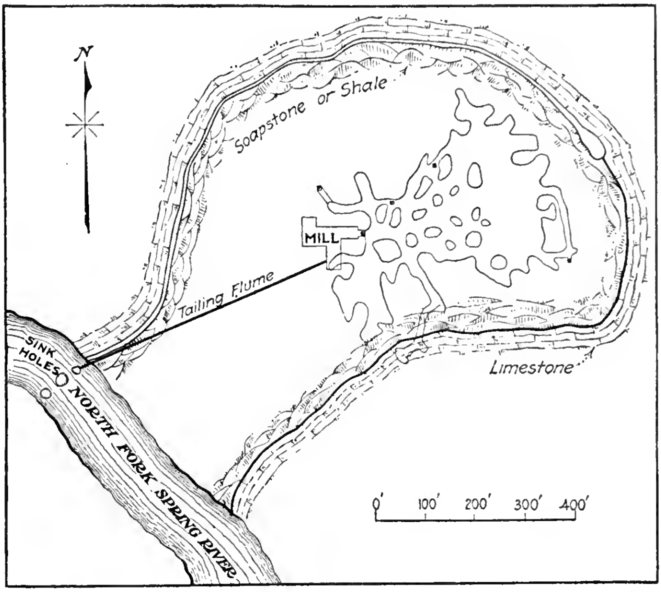


FIG. 433.—MAP SHOWING RELATION BETWEEN RIVER AND MINE.

sink holes. In two hours after the tailings had been turned into the holes the water had been cut down to the normal flow of 700 gal. per minute. When nearly filled, boulders were piled on the tailings up to the level of the bottom of the river to serve as ballast and keep the river from washing out the tailings. Three other breaks in the river at different periods were stopped in the same manner, and the river permanently choked off.

VENTILATING DEVICES

Pressure Ventilation in Cripple Creek (By S. A. Worcester).—The pressure system of ventilation consists of simply sealing or bulkheading the mine practically air-tight and forcing in air under what pressure is

necessary to expel mine gas through a fissured formation. Low pressures only are used, up to 3 oz. per square inch and are obtained by means of a jet or a fan. The pressure system is applied in the Cripple Creek district where many of the mines are seriously affected by the escape of gas, consisting largely of CO_2, from the rocks.

Some ingenuity and care are required in selecting the point of installation for the fan or jet and in shutting off free escape of the air through mine workings. Thus at the Conundrum mine, a short drift *GHT*, Fig. 434, connecting the main shaft with the Gold Hill tunnel, was first selected for the fan. It was found, however, that gas issued from the Gold Hill tunnel and mixed with the air which was being forced into the mine. The fan was therefore moved to a chamber *EB* cut in the Conundrum

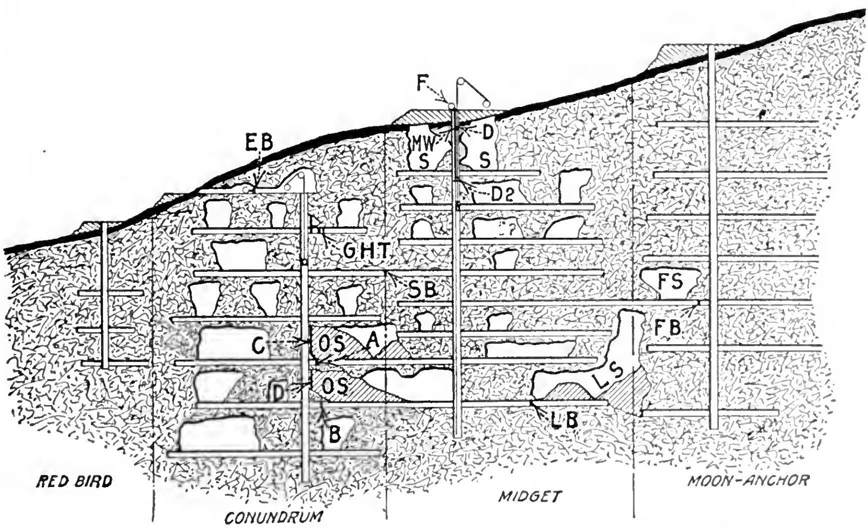


FIG. 434.—PRESSURE VENTILATION IN SEVERAL CONNECTED MINES.

adit at the entrance bulkhead. This fan has a 56-in. blast wheel, a 30-sq. in. discharge opening, runs at 340 r.p.m., is driven by a 15-hp. motor and operates for about eight and one-half hours daily. In the Midget mine a set of doors was first placed in the vertical shaft at *D*, 20 ft. below the shaft mouth and the manway *MW* was lined with 1-in. boards battened with canton flannel soaked in P. & B. paint. This made a wind trunk from the blower through the door frame bulkhead. The air was found to short-circuit through old stopes, which could not easily be shut off. Therefore the fan was moved to *D2* and the wind trunk extended to that point. The fan *F* is a converted steel-plate exhauster with one side inlet and is set directly over the manway. It has a 66-in. blast wheel and a 36-sq. in. discharge; it is driven at 400 r.p.m. by a 20-hp. motor.

At the Conundrum mine two worked-out stopes partly filled with fine rock connected with the Midget. The air passed freely through these, and the bulkheads *A* and *B* did not stop leakage to the Midget. Calking with oakum between the round lagging at *C* and *D* did not stop the leakage although it absorbed much P. & B. paint. Finally the entire front of these stopes at *C* and *D*, a distance of about 70 ft., was boarded over with 1-in. dressed boards, fitted carefully to the foot and hanging walls and battened with canton flannel strips well soaked in P. & B. paint. This made them practically air-tight. Where a level connects directly from one mine to the other, as at *SB*, a permanent bulkhead of 1-in. lumber is built, with the dressed side toward the pressure and two 3 × 8-in. girts behind, securely wedged in hitches. All loose rock is cleared away at the top, sides, and bottom down to solid, and swept clean. Boards are nailed upright and trimmed so as to fit closely the irregular rock surface. The rocks and the board are painted, and the canton-flannel strip painted on both sides is stuck over the crack and again painted. Nothing else equals P. & B. paint for this purpose. A doorway 20 × 24 in. is cut about 18 in. above the level; a trap door, 23 × 27 in., covers this opening. It must not fit between strips edgewise, or moisture might make it bind. It must lie flat against the bulkhead, with four thicknesses of canton flannel, nap side out, tacked on in strips 1½ in. wide around the edges where it rests on the bulkhead. A cheap door may be made without hinges and merely held by two buttons. This permits the passage of a man for the purpose of inspecting the bulkhead with a candle. A slight leak, when pressure is on, will give current enough to deflect a candle flame. Where a worked out stope, *LS*, partly filled with waste was found, passing from one property into the other, and where a bulkhead at the property line would be expensive, the stope, since it had no value, was simply shut off by placing a permanent bulkhead, *LB*, on the level. Leasers having ground which they desired to work in an adjoining mine, at *FS*, arranged with the Midget to ventilate their workings by placing a bulkhead at *FB*.

Fig. 435 shows the details of an entrance bulkhead for tunnels or drifts using an air-jet injector of crude form for supplying air. To erect it, proceed as follows: Select a place in solid ground, free from cracks or open fissures, and clean away all loose rock at the bottom, down to the solid, sweeping clean at the bottom, sides, and top. Set the 3 × 10-in. door posts 1, wedging them solidly in hitches, with the cap 2 spiked between them. Nail 1-in. boards on the front and back of the posts to a height within ¼ in. of the plate 3. Nail 1-in. boards horizontally on the front side of the posts and cap up to the top of the drift, trimming with a compass-saw the ends next to the rock to fit the rough surface closely, leaving no opening wider than ½-in. Make a 1:5 cement mortar with

about enough water to show on top after the mortar has settled a short time. Fill the bottom of the bulkhead to the height of the plate 3, which should be leveled, embedding many large clean rocks in the mortar, but taking care that they do not touch the form, as that causes voids. Embed the plate in wet mortar, tamping concrete under the edge at 4. The plate is usually 24 × ¼ × 36 in., and must fit closely between the posts. After the bottom is filled, nail 1-in. boards singly, smooth side out, on the inside of the frame, filling with mortar and rock as each piece is nailed, and making sure that no voids are left. As the height increases the pressure of mortar on the lower boards increases, and if the drift or tunnel is large, making these boards long, it will be necessary either to brace the unsupported ends or, better still, to run wires through from one side to the other, to prevent bulging. When approaching the top of the

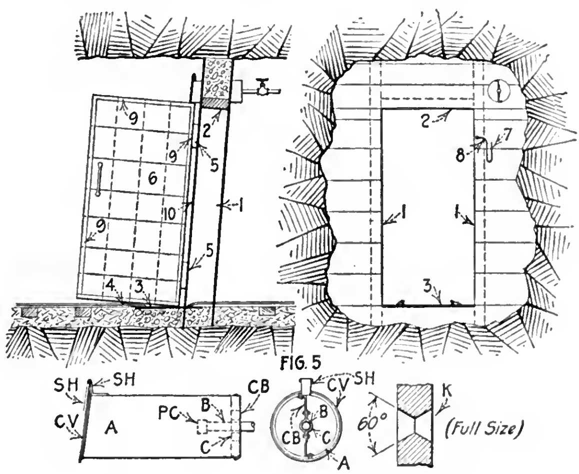


FIG. 435.—AIR-TIGHT DOOR AND AIR INJECTOR.

bulkhead, the mortar is made drier, and only small rocks, if any, are used in order to make a good joint and prevent undue shrinkage in drying. The edges where the bulkhead joins the rock should be gone over with clear cement when the bulkhead has set four or five days, and all shrinkage cracks closed, particularly around the top, where there is usually some settling. The door, 6, is made of two thicknesses of 1-in. clear-pine dressed boards. Two thicknesses of the heaviest canton flannel are placed between the boards and 2-in. nails are used close together and clinched on the pressure side. The boards next to the door frame are horizontal. To prevent twisting, the door should be nailed together on leveled horses or supports.

Care must also be taken in setting the bulkhead posts to have them "out of wind." The door must not fit between strips or in the frame, but

must lie flat on the bulkhead. The door laps the opening 1½ in. at the top and sides, and a canton-flannel strip of four thicknesses, 1½ in. wide, is tacked around the door at 9, where it closes against the 1-in. facing 10. A 5-in. 4-thick strip is doubled around the bottom edge of the door and tacked on both sides. All of these strips are laid with the nap outward and fastened with carpet tacks spaced about 1 in. apart. The track rails are cut off and the ends tapered down and curved so as to form horns, which rest on the plate, clearing the door. The bulkhead is inclined 1 in. per foot, so that the door swings clear of the plate and rails when opening. Ten-inch T-hinges are screwed to the door and to blocks 5, which are of the same thickness as the door screwed to the bulkhead. The pressure gage 7, a glass U-tube, is placed at a convenient height on the outside of the bulkhead and connected by a short rubber tube 8 with a nipple passing through the bulkhead. The glass tube is half filled with kerosene. If it is less than ¼ in. inside diameter, capillarity will impede its action. It is well to keep this pressure gage in good order, because it indicates any failure of pressure, with the accompanying danger from gas.

The air jet or injector, Fig. 435, includes a short 8-in. galvanized pipe *A*, into which the 1-in. compressed-air pipe *B* projects a short distance. Usually it is well to have at hand three different pipe caps, *PC*, with orifices about 3/32 in., 3/16 in. and 5/16 in. diameter. The smallest may be used for ventilating small workings or in good weather, and the others for heavier demands. The 3/32-in. jet seems to use about as much air as one hammer drill. A check valve, *CV*, of light tin is held by the small strap hinge, *SH*, anc has four thicknesses of canton flannel held by its turned edge, making a good joint with the wired edge of the pipe *A*. The cross-bar *C*B and the clamp *C* supporting the 1-in. pipe centrally are made of 1 × 3/16-in. bar, with ¼-in. stove bolts. The circular jet orifices are reamed as shown full size at *K*, giving a spreading discharge to the compressed air. Any tinner can make this air-jet outfit in a short time, and at small cost. This device is only recommended for temporary use of where compressed air is cheap.

A fan electrically driven is a more economical arrangement in most cases, especially where large volumes of air are needed. The check valve *CV* may be hung up, so as not to impede the air current during the shift, but is dropped to its seat when the jet is shut off and the door is left closed, with the effect of impeding appreciably the entrance of mine gas to the workings. The end of the pipe *A* is inclined so that the check valve shuts by gravity.

The doors used at the Midget vertical shaft, where a skip is used for hoisting, are made of double 2-in. clear pine, with two thicknesses of canton flannel between, and have forged hinges. They are notched to clear the guides and have all seating or meeting edges stripped with four thick-

nesses of flannel, tacked closely, nap side out. The rod *R*, Fig. 436, with the links *L*, opens and closes the doors together. The spring *S* assists in starting the doors from the vertical position toward closing, and the steel bell-cord *BC* passes up the shaft, over a sheave to the lever *L* with fulcrum at *F*, and is placed within easy reach of the hoist engineer. The weight *W* nearly balances the doors and spring. The parts are held in the open or closed position by hooking the lever *L* over the stationary block *B*. The weight of these doors and connections is sufficient to

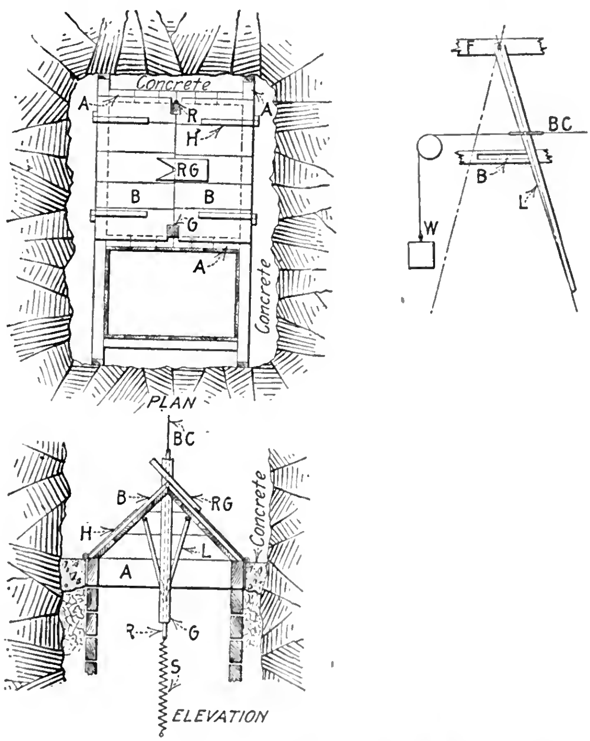


FIG. 436.—DOOR ARRANGEMENT IN MIDGET SHAFT.

resist the pressure used at this mine. The rope guide *RG* is bolted to the door and is easily renewable when worn. It directs the rope into the center groove when the doors are closed, preventing it from catching between the meeting edges.

When installing shaft doors at the Little Nell mine, Fig. 437, a place was chosen in solid rock 15 ft. below the shaft collar, and enough of the shaft timbering was removed to make room for the frame *A* and for swinging the doors *B* clear of the shaft. The rock was swept clean on all sides, to give a good joint for concrete. The sides of the frame *A*

extend through the manway, giving sills for the bulkhead of 1-in. boards. The cracks are battened with the heaviest canton flannel, well soaked and stuck on the freshly painted surface with P. & B. paint, then painted on the back. The frame *A*, being firmly wedged in place and leveled, is concreted all around, using the mortar described in the foregoing. Many large clean rocks are embedded in the mortar, taking care to leave no voids and making the top about 2 in. of clear mortar, with no rocks. The doors *B* are made of 1-in. clear dressed boards, crossed, with two thicknesses of canton flannel between and 2-in. clinched nails spaced closely. They should be free from twist and lie flat on the frame. The frame should be dressed where the doors seat. Canton-flannel strips *FS*, about 1½ in. wide, in four thicknesses, are tacked around the three edges of the doors where they rest on the frame and on the under side of the

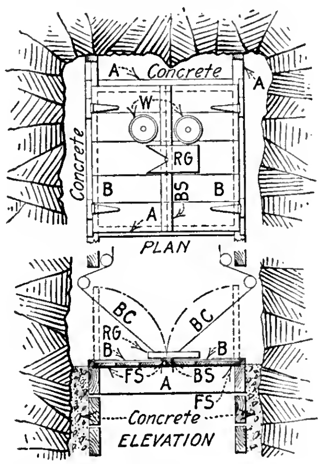


FIG. 437.—DOOR IN LITTLE NELL SHAFT.

batten strip *BS*. The doors are notched at the center to fit the hoisting rope nicely, and the rope guide *RG*, of hard pine, is screwed to the door. The lower boards run lengthwise. Ten-inch T-hinges are used. The weights *WW*, in this case worn-out sheave-wheels, were determined by experiment after pressure on the door, and are bolted to the door. The batten strip is fast to the door having the rope guide, and this door must be opened first and closed last to prevent interference. Cotton bell-cords *BC* are fastened to staples near the edges of the doors and are led by pulleys to handles with hooks within easy reach of the engineer. The doors must not fit edgewise but lie flat, or moisture will make them troublesome.

Many fixed bulkheads have been built in shafts and raises by simply stulling and lagging the opening and packing a few inches of black surface

soil on top of the lagging. Ordinary mine rock, however fine, seems to leak very much, but black surface soil makes an air-tight stopping when well packed.

Automatic Door for Ventilation Control (By H. S. Gieser).—At the Jupiter, which is one of the deep-level mines in the Germiston area of the East Rand, there are two shafts, the Howard and the Catlin, both of which are part vertical and part inclined. The bottom of the Catlin is 5040 ft. vertically below the surface and at these great depths the heat is excessive. Mechanical ventilation is employed and to guide the air through the stopes, automatically operated doors are provided on the levels, approximately as shown in Fig. 438. The sliding door is made in two leaves A, which slide laterally by means of the sheaves B on the 1-in. inclined round irons C. The doors are made of a layer of 1-in. vertical boards and a layer 1¼-in. horizontal boards and are of the dimensions shown. The sheaves are 6 in. in diameter and carry the doors by iron straps D, 3 × 28 in. The round irons are carried on timber pieces E, 3 × 3 in. by 4 ft. 3 in., inclined from the center of the drift at an angle of 10°. As the doors are forced open, the sheaves roll out and up on the round irons against gravity and, when released again by the passage of the car, they slide together by their own weight.

The doors are actuated by the four angle irons F, set two on each side so that cars from either direction will open the doors. Each angle has one end pivoted on the inner edge of the door at G, and the other end slotted and moving on a fixed pin at H. They are set at a height to catch a car body, and on each side of the door one rail is slightly higher than the other. The car entering between them forces them apart and opens the doors, and as it passes, allows them to close in a similar manner. The angles are 2½ × 2½ in. and are 10 ft. long, set with one outer side up and one toward the center of the drift. The slot at H is ⅞ × 4½ in. The drift, approximately 7 ft. wide, is closed for the most part by a 1-in. wood brattice I, only enough opening, 3 ft. 4 in., being left for the door to cover, as will admit a car. This opening is contained in a frame of 6 × 6-in. stuff J, to which, as well as to slanting 3 × 6-in. pieces K, the 1-in. brattice is fastened. The pins on which the angles slide are carried by a frame L of 6 × 6-in. material.

Ventilating Pipes.—A useful novelty in ventilating equipment, especially adapted to the wide connecting drives, raises and winzes of the district, is described by H. Bottomly in an annual report of the Department of Mines and Industries of South Africa. The device consists of ⅛-in. steel plates, 6 × 4 ft., bent to U-shapes, 18 in. across and 18 in. deep. The sections are inverted and laid on the floor of the drive, loose ground being placed against the contact with the rock to insure a tight joint. Sections are machine rolled and are capable of being bolted to-

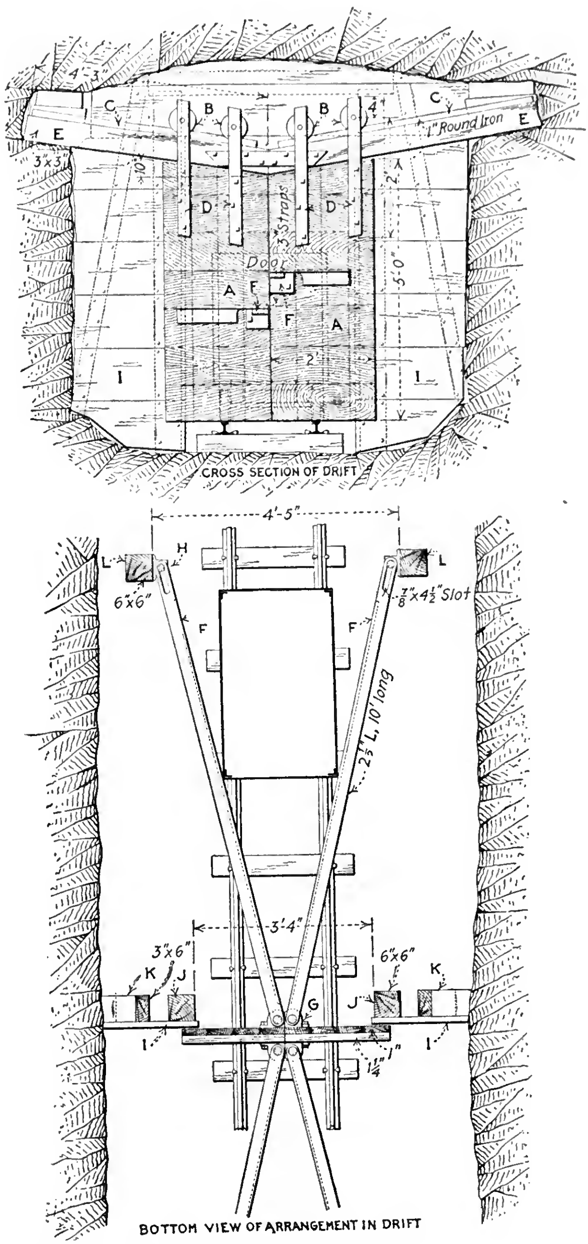


FIG. 438.—DOOR TO CONTROL VENTILATION THROUGH DRIFT.

gether in a tight joint, assisted by a gasket of tarred cloth. Area available in these pipes is 2 sq. ft., which is far superior to the 10-in. galvanized-iron pipes in common use, and their comparative immunity from damage places them above anything else in use for the purpose at the present time. The manager of the State Mines, Graham Bell, the inventor of the system, has tried galvanized-iron pipes at 10d. per foot, and corrugated-iron bratticing along the main connection drives at a cost of 5s. per foot, but has given both up for the steel pipes now in use, which cost 3s. 7d. per running foot and require practically no maintenance.

Tunnel and Level Ventilation.—The ventilation of workings reached through long tunnels can probably be accomplished by no simpler means than that employed in one of the older Mexican mines. This mine was worked through a tunnel over a mile long. To establish a system of

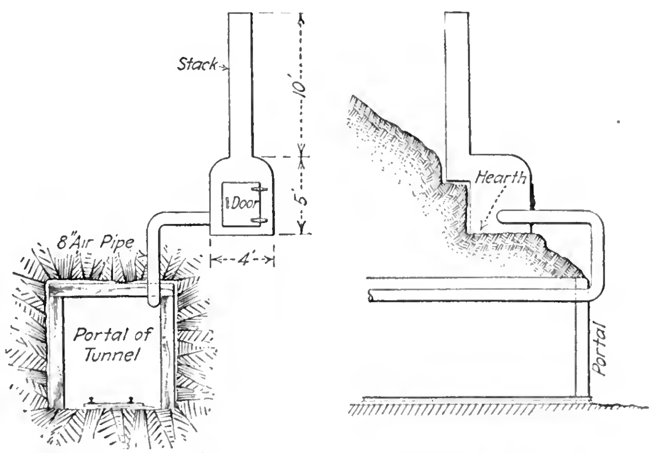


FIG. 439.—STOVE FOR SUCKING GASES FROM TUNNEL.

ventilation for reaching the workings inside, a deep channel was cut in the floor of the tunnel, and this channel was covered with flat stone, or *loza*, set in cement. In this way the water conduit was made to serve also as an airway. The water flow assisted in drawing out the air. Issuing from the tunnel the water conduit was connected with a length of nearly vertical pipe down which the water ran, acting as an aspirator and setting up a good circulation of air in the mine.

Adobe Stove for Tunnel Ventilation (By T. Swift).—Fig. 439 shows a modification of the stove ventilator, as used in Mexico. This stove was built of adobe bricks and was rather large, 4 × 4 ft. square, 5 ft. high, with a stack 10 ft. high. It was built outside of the tunnel and above the portal. The air pipe was run from the last set of timber near the face of the tunnel and entered the stove at the top of the hearth. A roaring wood fire would suck the powder smoke from the 400-ft. tunnel after

shooting, in 10 or 15 min. The arrangement seems to require only that the top of the hearth, where the pipe enters, be above the elevation of the pipe at the face. In cases when the draft does not start readily and "*la estufa no quiere*" a damper in the pipe at the stove will overcome the difficulty, if kept closed until the fire is going briskly and then opened.

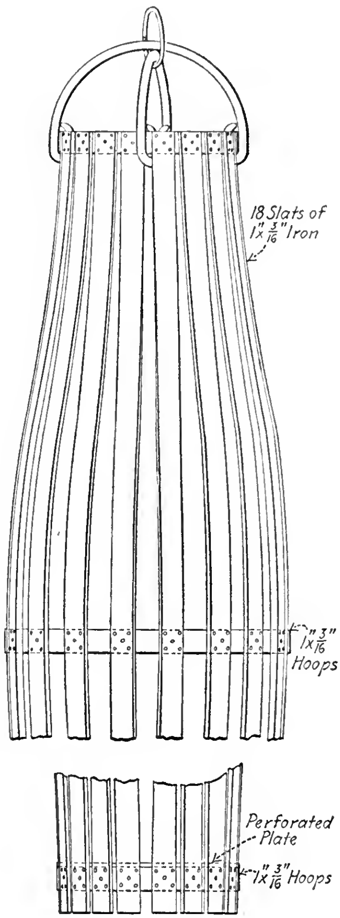


FIG. 440.—CHARCOAL POT FOR VENTILATING SHAFT.

This arrangement was not scientifically built or investigated, but it worked well and is handy where natural ventilation or compressed air are lacking.

Charcoal Pot for Ventilating Shaft.—During the course of the sinking operations at the No. 6 shaft on the property of the St. Louis Smelting & Refining Co., in the Flat River district of Missouri, a charcoal fire was

used to induce ventilation and dissipate rapidly the fumes from blasting. The fire was built in an iron basket such as is illustrated in Fig. 440. This basket, filled with burning coals, is lowered to the bottom of the shaft immediately after blasting. The heating of one side of the shaft causes an upward draft and a down draft on the other side.

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